Barite

Asbestos

Magnesite

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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. L.

CONTAINING THE PAPERS AND DISCUSSIONS OF THE PITTSBURGH MEETING, OCTOBER, 1914, AND PAPERS AND DISCUSSIONS OF THE NEW YORK MEETING, FEBRUARY, 1914, RELATING TO IRON AND STEEL

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PREFACE.

This volume, the third of the three volumes containing the papers and discussions contributed to the Institute during the year 1914, comprises papers presented at the Pittsburgh meeting, October, 1914, and the discussion thereon. In addition, papers and discussions relating to iron and steel which were presented at the New York meeting of February, 1914, but omitted from Vol. XLVIII for lack of space, are included in this volume.

CONTENTS.

PROCEEDINGS

Pittsburgh Meeting, October, 1914 vii

PAPERS

PAGE

Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces. By A. N. DIEHL (with Discussion) 3

Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes. By HEINRICH J. FREYN (with Discussion) 56

Notes on Blast-Furnace Operation with a Turbo Blower. By S. G. VALENTINE (with Discussion) 90

Turbo Blowers for Blast-Furnace Blowing. By RICHARD H. RICE (with Discussion) 104

The Iron Industry in Brazil. By E. C. HARDER (with Discussion) 143

Pig Steel from Ore in the Electric Furnace. By ROBERT M. KEENEY (with Discussion) 161

Valuation of Iron Mines (FINLAY, *Trans.*, xlv, 282). (Discussion by E. E. WHITE) 188

The Reserves of Iron Ore for the United States. By JOHN BIRKINBINE . . . 197

The Need of Uniform Methods of Sampling Lake Superior Iron Ore. By C. B. MURRAY (with Discussion) 204

Notes on Conservation of Lake Superior Iron Ores. By C. K. LEITH (with Discussion) 231

Notes on an Iron-Ore Deposit near Hong-Kong, China. By C. M. WELD . . 236

The Plant of the Duplex Process for Making Steel. By J. K. FURST (with Discussion) 246

Sound Ingots. By SIR ROBERT HADFIELD 270

Rolled Steel Roll Shells. By JAMES C. H. FERGUSON 290

Finishing Temperatures and Properties of Rails. By GEORGE K. BURGESS, J. J. CROWE, H. S. RAWDON, and R. G. WALTENBERG (with Discussion) . . 302

Manganese-Steel Rails. By SIR ROBERT HADFIELD (with Discussion) . . . 327

The American Steel-Rail Situation. By ROBERT W. HUNT 340

The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements. By J. E. JOHNSON, JR. (with Discussion) 344

The Surface Decarbonization of Tool Steel. By J. V. EMMONS (with Discussion) . 405

The Heat Treatment of Steel Castings. By C. D. YOUNG, O. D. A. PEASE, and C. H. STRAND (with Discussion) 424

Manganese Steel, with Especial Reference to the Relation of Physical Properties to Microstructure and Critical Ranges. By W. S. POTTER (with Discussion) 437

Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel. By PROF. B. HOPKINSON and SIR ROBERT HADFIELD (with Discussion) 476

Manganese Steel and the Allotropic Theory. By ALBERT SAUVEUR (with Discussion) 501

PAGE

Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron. By ALBERT SAUVEUR (with Discussion) 515
Notes on the Plastic Deformation of Steel During Overstrain. By HENRY M. HOWE and ARTHUR G. LEVY (with Discussion) 532
Investigations of Coal-Dust Explosions. By GEORGE S. RICE (with Discussion). 552
Coal-Dust Explosion Investigations. By J. TAFFANEL 588
Coal-Mine Explosions Caused by Gas or Dust. By HOWARD N. EAVENSON (with Discussion) 594
The Appraisal of Coal Lands for Taxation. By H. M. CHANCE (with Discussion) 625
The Pittsburgh Coal Field in Western Pennsylvania. By H. A. KUHN . . . 640
The Book Cliffs Coal Field, Utah. By ROBERT S. LEWIS 658
An Aerial Tramway for Mining Cliff Coal. By ARTHUR E. GIBSON 679
Tin and Coal Deposits of the Fu Chuan District, China. By M. B. YUNG (with Discussion) 689
Steep Pitch Mining of Thick Coal Veins. By W. G. WHILDIN 698
Shot Firing in Coal Mines by Electric Circuit from Surface. By GEORGE S. RICE and H. H. CLARK (with Discussion) 723
A New Safety Detonating Fuse. By HARRISON SOUDER (with Discussion) . . 738
The Safety Movement in the Lake Superior Iron Region. By EDWIN HIGGINS. 755
Gasoline Locomotives in Relation to the Health of Miners. By O. P. HOOD (with Discussion) 771
A Test of Centrifugal Motor-Driven Pumps. By S. S. RUMSEY and W. F. SCHWEDES (with Discussion) 779
Refining Petroleum by Liquefied Sulphur Dioxide. By L. EDELEANU (with Discussion) 809
The Capillary Concentration of Gas and Oil. By C. W. WASHBURNE (with Discussion) 829
The Oil Fields of Mexico. By ESEQUIEL ORDOÑES (with Discussion) 859
Gas and Oil Wells through Coal Seams. (A Discussion) 870
Chlorides in Oil-Field Waters (*Trans.*, xlviii, 687 to 694). By C. W. WASHBURNE 883
The Occurrence, Preparation, and Use of Magnesite. By L. C. MORGANROTH (with Discussion) 890
The Origin, Mining and Preparation of Phosphate Rock. By E. H. SELLARDS . 901
Tennessee Phosphate Practice. By JAMES A. BARR 917
Salt Making by Solar Evaporation. By W. C. PHALEN (with Discussion) . . 934
Asbestos in Southern Quebec. By JOHN A. DRESSER (with Discussion) . . . 954
Asbestos Deposits of Georgia. By OLIVER B. HOPKINS (with Discussion) . . 964
Quarrying Shale by the Tunnel System. By DWIGHT T. FARNHAM (with Discussion) 974
Barytes as a Paint Pigment. By H. A. GARDNER and G. B. HECKEL 983

Proceedings of the One Hundred and Ninth Meeting, Pittsburgh, Pa., October, 1914

LOCAL COMMITTEES

Executive Committee.—Samuel A. Taylor, *Chairman;* C. F. W. Rys, *Secretary;* O. P. Hood, F. C. Phillips, George S. Rice, Kenneth Seaver, J. S. Unger.

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Entertainment Committee.—W. L. Scaife, *Chairman;* John W. Boileau, W. E. Fohl, E. K. Hiles, W. W. Keefer, W. G. Wilkins.

TECHNICAL SESSIONS

Thursday Afternoon, Oct. 8, 1914.—The opening session was called to order in the Lecture Hall of the Carnegie Institute at 2:30 p.m., with Samuel A. Taylor presiding. An address of welcome was delivered by Dr. W. J. Holland, after which the following papers were presented by their authors or authors' representatives:

The Occurrence, Preparation, and Use of Magnesite, by L. C. Morganroth. Discussed by David T. Day, Thomas T. Read and E. Gybbon Spilsbury.

The Iron Industry in Brazil, by E. C. Harder. Discussed by I. C. White.

Investigations of Coal-Dust Explosions, by George S. Rice. Discussed by William Griffith.

The Appraisal of Coal Lands for Taxation, by H. M. Chance. Discussed by R. V. Norris, William Griffith, Samuel A. Taylor, R. Dawson Hall, H. N. Eavenson.

Thursday Evening, Oct. 8, 1914.—An informal reception was held in the Lecture Room of the Carnegie Institute preceded by an illustrated lecture at 8:30 p.m. on Ancient Methods of Manufacture of Iron in China, by Thomas T. Read. This was followed by moving pictures on Safety Methods in Mining and Metallurgy, shown by representatives of the U. S. Bureau of Mines.

Friday Morning, Oct. 9, 1914.—A session was held at 9:30 a.m.,

under the auspices of the Iron and Steel Committee, Prof. Albert Sauveur presiding. The following papers were presented by their authors or authors' representatives:

The Reserves of Iron Ore for the United States, by John Birkinbine.

Finishing Temperatures and Properties of Rails, by G. K. Burgess, J. J. Crowe, H. S. Rawdon, and R. G. Waltenberg. Discussed by P. H. Dudley (in writing), R. Trimble (in writing), William R. Webster (in writing), George B. Waterhouse, M. H. Wickhorst, James Aston (in writing), E. F. Kenney, A. W. Gibbs, Leonard Waldo (in writing), Albert Sauveur, G. K. Burgess.

The Plant of the Duplex Process for Making Steel, by J. K. Furst. Discussed by W. McA. Johnson (in writing), Arthur G. McKee, Henry D. Hibbard, and Bradley Stoughton.

Manganese Steel and the Allotropic Theory, by Albert Sauveur. Discussed by H. M. Boylston (in writing), George K. Burgess (in writing), Albert Sauveur.

Friday Morning, Oct. 9, 1914.—A session was held at 9:30 a.m. under the auspices of the Committee on Coal and Coke, in the Assembly Hall of the University of Pittsburgh, H. H. Stoek and R. V. Norris presiding. The following papers were presented by their authors or authors' representatives:

Coal-Dust Explosion Investigations, by J. Taffanel.

Coal-Mine Explosions caused by Gas or Dust, by H. N. Eavenson. Discussed by George S. Rice, C. M. Young, H. H. Stoek, Howard N. Eavenson, R. Dawson Hall, William Griffith, W. H. Grady, R. V. Norris, E. B. Wilson, F. Z. Schellenberg, Samuel A. Taylor, Richard Peters, Jr.

The Pittsburgh Coal Field in Western Pennsylvania, by H. A. Kuhn. Read by title only.

The Book Cliffs Coal Fields, Utah, by R. S. Lewis. Read by title only.

An Aerial Tramway for Mining Cliff Coal, by A. E. Gibson. Discussed by Samuel A. Taylor.

Tin and Coal Deposits of the Fu Chuan District, China, by M. B. Yung. Discussed by Thomas T. Read and George S. Rice.

* The Manufacture of Coke, by J. P. K. Miller, Jr. Discussed by I. C. White, Thomas T. Read, William H. Blauvelt, Samuel A. Taylor.

Friday Morning, Oct. 9, 1914.—A session was held in the parlor of the Hotel Schenley under the auspices of the Committee on Non-Metallic Minerals, Heinrich Ries presiding. The following papers were read by their authors or authors' representatives:

Salt Making by Solar Evaporation, by W. C. Phalen. Discussed by E. Gybbon Spilsbury, Waldemar Lindgren, David T. Day, and Heinrich Ries.

* Not included in this volume.

Asbestos in Southern Quebec, by J. A. Dresser.

Asbestos Deposits of Georgia, by O. B. Hopkins.

These two papers were discussed jointly by E. Gybbon Spilsbury, David T. Day, J. A. Dresser, Waldemar Lindgren, H. A. Wheeler, and Heinrich Ries.

Quarrying Shale by the Tunnel System, by D. T. Farnham. Discussed by David T. Day, Heinrich Ries, and Donald W. Ross.

The following papers were read by title:

The Origin, Mining, and Preparation of Phosphate Rock, by E. H. Sellards.

Tennessee Phosphate Practice, by J. A. Barr.

Barytes as a Paint Pigment, by H. A. Gardner and G. B. Heckel.

Saturday Morning, Oct. 10, 1914.—A session on Petroleum and Gas was held at 9:30 a.m., under the auspices of the Committee on Petroleum and Gas, Dr. David T. Day presiding. Reports were presented by the Chairmen of the Sub-Committees of the Committee on Petroleum and Gas, and the following papers were presented by their authors or authors' representatives:

The Oil Fields of Mexico, by Ezequiel Ordoñez. Discussed by David T. Day, William N. Best, Philip W. Henry.

The Capillary Concentration of Gas and Oil, by C. W. Washburne. Discussed by William N. Best, Leonard Waldo, Roswell H. Johnson, David T. Day, H. M. Chance, H. A. Wheeler, G. A. Burrell, F. W. Bushong.

This was followed by a general discussion on Gas and Oil Wells through Coal Seams. Discussed by George S. Rice, David T. Day, W. E. Fohl, M. B. Layton, R. Dawson Hall, O. P. Hood, G. A. Burrell, F. H. Ramsey.

The following paper was read by title:

Refining Petroleum by Liquefied Sulphur Dioxide, by L. Edeleanu.

Saturday Morning, Oct. 10, 1914.—A session on Electricity and Miscellaneous Mining Topics was held at 9:30 a.m. under the auspices of the Committee on the Use of Electricity in Mines, William Kelly presiding. The following papers were presented by their authors or authors' representatives:

A Test of Centrifugal Motor-Driven Mine Pumps, by S. S. Rumsey and W. F. Schwedes (read by title only). Discussed by K. A. Pauly, William Kelly, George S. Rice, O. P. Hood, H. H. Stoek, W. A. Thomas, R. V. Norris, Edwin Ludlow.

Gasoline Locomotives in Relation to the Health of Miners, by O. P. Hood. Discussed by R. V. Norris, George S. Rice, Samuel A. Taylor, W. G. Whildin, O. P. Hood.

Shot-Firing in Coal Mines by Electric Circuit from the Surface, by George S. Rice and H. H. Clark. Discussed by Norman V. Breth,

George S. Rice, T. H. O'Brien, William Kelly, Samuel A. Taylor, O. P. Hood, H. H. Stoek, H. H. Clark, C. M. Means.

Saturday Morning, Oct. 10, 1914.—A session on Iron and Steel was held under the auspices of the Iron and Steel Committee, Prof. Albert Sauveur presiding.

The following papers were presented by their authors or authors' representatives:

A New Safety Detonating Fuse, by Harrison Souder.

Rolled Steel Roll Shells, by James C. H. Ferguson (read by title).

Surface Decarbonization of Tool Steel, by J. V. Emmons. Discussed by Albert Sauveur, Bradley Stoughton, E. Gybbon Spilsbury, H. O. Hofman, J. V. Emmons.

Turbo Blowers for Blast-Furnace Blowing, by Richard H. Rice. Discussed by S. G. Valentine (in writing), W. McA. Johnson (in writing), J. E. Johnson, Jr., L. Iversen, Karl Nibecker, W. Trinks (in writing), Otto Banner (in writing), H. S. Braman, S. M. Marshall (in writing), Arthur G. McKee, Richard H. Rice.

The Safety Movement in the Lake Superior Iron Region, by Edwin Higgins. Discussed by Howard N. Eavenson.

SOCIAL FEATURES OF THE MEETING AND VISITS

Headquarters and Registration.—The Headquarters was at the Hotel Schenley, and opportunities for registration were afforded throughout the entire meeting in the lobby of the hotel.

Dinner.—A dinner was held in the main dining room of the Hotel Schenley at 7 :30 p.m. on Friday, Oct. 9, 1914, which was attended by a large number of members and lady guests. By special arrangements business suits were worn by the men.

Excursions.—On Thursday evening, Oct. 8, 1914, opportunity was given for members to visit the Alleghany observatory to view the new refractory, which had just been installed in connection with the Thaw Memorial Telescope. Some of the members also availed themselves of the invitations to visit the Carnegie Technical Schools, the University of Pittsburgh, and the Carnegie Museum to view the radium exhibit.

On the afternoon of Friday, Oct. 9, 1914, the following excursions were made by different parties of members and guests, all arrangements having been made in advance for personally conducting the visitors to the points mentioned:

The Homestake Steel Works of the Carnegie Steel Co.

McKeesport Plant of the National Tube Co.

Brick Factories of the Harbison-Walker Refractories Co.

Experimental Mine of the U. S. Bureau of Mines at Bruceton, Pa., to witness a coal-dust explosion, and the stopping of same at certain

predetermined points in the mine entries by the employment of rock-dust barriers. A demonstration was given of the use of the cement gun, putting a fire-proof, tight coating over a wooden stopping. The cement gun has been used in the experimental mine for coating certain parts of the mine entries without the use of wire mesh, the thin coating adhering directly to the coal and shale.

On Saturday afternoon, Oct. 10, 1914, the visitors were conducted through the laboratories of the U. S. Bureau of Mines at Pittsburgh, Pa., where a demonstration was given of the different researches being conducted by the Bureau of Mines.

During the time of the meetings the ladies were given an opportunity to inspect the works of H. J. Heinz Co.

PAPERS

Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces

BY A. N. DIEHL, DUQUESNE, PA.

(New York Meeting, February, 1914)

CONTENTS

PAGE

History . . . 4
Constitution of Blast-Furnace Gas . . . 4
True Gas . . . 4
Carbon Monoxide and Dioxide . . . 4
Hydrogen . . . 5
Hydrocarbons . . . 5
Moisture . . . 6
Solids . . . 8
Mechanically Carried Flue Dust . . . 8
Chemical and Physical Characteristics . . . 8
Settling Properties . . . 10
Quantity Carried by Furnace Gas . . . 12
Nature and Quantity Passing through the System . . . 14
Description of Plant . . . 14
Scrubbers . . . 18
Fans . . . 21
Theisens . . . 22
Settling Basins . . . 23
Operation of Plant . . . 25
Daily Observations . . . 30
Sampling Gas for Analysis . . . 31
Conducting and Reporting Tests . . . 31
Moisture and Dust Determinations . . . 34
Raw vs. Clean Gas . . . 36
Use of Raw Gas . . . 36
Use of Clean Gas . . . 36
Use of Refined Gas . . . 36
Comparison of Systems . . . 40
Cost of Gas Cleaning, Year 1912 . . . 41
Conclusion . . . 46

It is the object of this paper (1) to deal with the elements in blast-furnace gas from the standpoint of their importance, and the part they are to play in future consumption, and (2) to give detailed information about the construction and operation of a plant designed among the first in this country for washing gas on a large scale.

History

The conservation and use of blast-furnace gas dates back to the earlier history of pig-iron manufacture. Before this period all the gas was wasted, but the advent of the hot-blast stove and steam blowing engines brought a demand for additional fuel, and the natural source of this was waste gas at the tunnel heads. The regulation of the intensity of the hearth temperature by the preheating of the air has been discussed on many occasions, and the value of the waste gas as a substitute for coal is a definite problem, so only a passing mention may be made of these applications, each being a subject requiring a separate study. The principal fuel in these instances is carbon monoxide together with small percentages of hydrogen and hydrocarbons. The heated air, entering at the tuyères and encountering a great excess of incandescent carbon, combines with sufficient carbon to form the lower carbon oxide and leaves the tuyère area in this condition at a temperature of 3,100° to 3,200° F. This hot, unsaturated gas containing 38 per cent. monoxide passes upward through the stock, giving up its sensible heat, and is partly oxidized to CO_2 by the ore until it leaves the top with an average composition of 12 per cent. CO_2 and 25 per cent. CO. The moisture in the blast will contribute about 3.5 per cent. of hydrogen by its decomposition, and the evaporation of the water in the stock also accounts for about 35 grains of moisture per cubic foot of gas. The hydrocarbons come from the volatile matter in the coke. The gas issues from the top at a velocity varying with the volume of air blown and the size of the tops and outlets, and its temperature is usually from 300° to 400° F. Its constitution also varies with the fuel practice of the furnace. The problem of handling the gas would be a comparatively easy one if the gas leaving the furnace did not carry along mechanically a considerable quantity of what is known as flue dust, as well as a material which afterward sublimes and is extremely hard to remove.

Constitution of Blast-Furnace Gas

Blast-furnace gas can then be considered as composed of: (1) true gas, (2) moisture, and (3) solids (consisting of (*a*) solids, and (*b*) volatile material which sublimes or is soluble in washing water).

Table I shows the average analysis of clean gas for engine purposes at the Duquesne Works for the year 1912.

True Gas

Carbon Monoxide and Dioxide.—As mentioned in the preceding paragraph, the gas relations of CO to CO_2 vary with furnace conditions and practice.

TABLE I.—*Analysis of Blast-Furnace Gas Delivered to the Gas Engines*

Year 1912

Month	Percentage by Volume										Specific Gravity
	CO_2	O_2	CO	CH_4	H_2	N_2	CO/CO_2	B.t.u. Gross	B.t.u. Net	Weight per Cubic Foot Pounds	
Jan.....	12.3	0	25.6	0	3.5	58.6	2.08	94.4	92.7	0.076636	1.00493
Feb.....	12.2	0	25.8	0	3.4	58.6	2.11	94.8	93.1	0.076661	1.00526
Mar.....	12.1	0	25.7	0	3.2	59.0	2.12	93.8	92.2	0.07676	1.00652
Apr.....	12.6	0	25.3	0	3.9	58.2	2.01	94.8	92.8	0.07649	1.00303
May.....	12.4	0	25.7	0	4.3	57.5	2.07	97.4	95.2	0.07614	0.99844
June.....	12.5	0	25.8	0	4.6	57.1	2.06	98.7	96.4	0.07596	0.99608
July.....	12.1	0	26.0	0	4.7	57.1	2.10	99.8	97.5	0.07566	0.99208
Aug.....	12.0	0	26.3	0	4.2	57.5	2.20	99.0	96.9	0.07602	0.99685
Sept.....	12.2	0	26.0	0	4.5	57.3	2.30	99.0	96.8	0.07592	0.99551
Oct.....	12.1	0	26.0	0	3.8	58.1	2.15	96.7	94.8	0.07635	1.00118
Nov.....	12.5	0	25.6	0	3.6	58.3	2.05	94.8	93.0	0.07666	1.00525
Dec.....	12.5	0	25.4	0	3.4	58.7	2.03	93.5	91.8	0.07679	1.00595
Avg.....	12.4	0	25.6	0	3.9	58.1	2.06	95.8	93.8	0.07627	1.00013

Period over which samples were taken averaged 7 hr. 30 min. (approx.).
Calorific value and weight per cu. ft. of gas at 62° F. and 30 in. Hg.
Specific gravity referred to air.

Hydrogen.—The amount of hydrogen varies with the moisture in the blast and the moisture which is decomposed from the water in the stock if, by a slip, the incandescent coke comes in contact with the wet stock. An example of this variation in hydrogen content is given from an experience during a slip as follows:

It was noticed that, following heavy slips on No. 6 furnace, the engines back fired badly. These back fires followed in the length of time occupied by the gas in passing through the system from the furnace to the engines. Several sample tubes were placed in readiness and, on a similar slip, gas samples were taken; the results are given on p. 6. Water to the extent of 8 per cent. of the ore weight was used at this time. These samples were checked three times in the laboratory. The engines back fired and prematured very badly during this slip.

Hydrocarbons.—The amount of hydrocarbons depends on the volatile matter in the coke; some are soluble in the washing water. An analysis of the water of condensation from the gas mains shows that, without a doubt, other organic matter was carried away in the scrubbing. A ferro-cyanide coloration is noticeable on the bottom of the scrubbers, although analysis will not show it.

Analyses of Blast-Furnace Gas taken at the Inlet to No. 9 Scrubber during a Slip on No. 6 Furnace

	Percentage by Volume		Percentage by Volume
I		II[a]	
Carbon dioxide	12.9	Carbon dioxide	16.8
Oxygen	0.1	Carbon monoxide	23.8
Carbon monoxide	19.8	Methane	0.2
Methane	1.6	Hydrogen	9.2
Hydrogen	38.8	Nitrogen	50.0
Nitrogen	26.8		
B.t.u. (62° F. and 30 in. Hg), 209.5		B.t.u. (62° F. and 30 in. Hg), 110.8.	

[a] Taken immediately after No. I and at the same place.

Water of Condensation.—This water has a strong nauseating smell and taste. It is clear and colorless, with the following analysis, after filtering off 2.9 parts per 100,000 of ferric oxide, Fe_2O_3:

	Parts per 100,000
Total solids	467
Inorganic solids	213
Organic solids	254

Iron is combined with an organic acid, is oxidized by air and precipitated. No evidence of cyanides was observed.

Moisture

The amount of moisture in the raw gas varies with the water in the stock. The use of Mesabi ore has made the addition of water a necessity in most cases of hard-driven furnaces in order to prevent dust being blown over, and this water, of course, is evaporated and carried by the gas. The effect of the water in the gas is shown in the following tables and calculations for different conditions and assumes the use of gas free from dust.

Inspection of Table II will disclose the fact that, per pound of dry fuel gas consumed, clean gas at 70° F. moisture saturated at 70° F. gives an available heat of 79.51 per cent. products of combustion cooled to 600° F. Raw gas at 400° F. and 35 grains moisture follows with available heat of 77.03 per cent. products of combustion cooled to 600° F. Partly washed and cooled gas at 125° F. gives an available heat supply of 74.33 per cent. products of combustion cooled to 600° F.

From a calculated thermal standpoint, we conclude, as evidenced by the table under inspection, that washed gas cooled to 70° F. gives the largest percentage of available heat. This is followed by hot un-

TABLE II.—*Heat Available per Pound of Dry Blast-Furnace Gas Under Various Conditions of Initial Temperature and Moisture Content. Combustion Air Constant Condition*

	Washed Blast-Furnace Gas at 70° F. with Moisture Saturated at 70° F. Combustion Air at 70° F. with Moisture Saturated at 57° F.			Washed Blast-Furnace Gas at 125° F. with Moisture Saturated at 125° F. Combustion Air at 70° F. with Moisture Saturated at 57° F.			Unwashed Blast-Furnace Gas at 400° F. with 35 Grains Moisture per cu. ft. Gas at 62° F. and 30 in. Hg. Combustion Air at 70° F. Moisture Saturated at 57° F.		
	Products of Combustion Cooled to 400° F.	Products of Combustion Cooled to 500° F.	Products of Combustion Cooled to 600° F.	Products of Combustion Cooled to 400° F.	Products of Combustion Cooled to 500° F.	Products of Combustion Cooled to 600° F.	Products of Combustion Cooled to 400° F.	Products of Combustion Cooled to 500° F.	Products of Combustion Cooled to 600° F.
TOTAL HEAT EXPENDED ABOVE 32° F.									
Fuel Gas.									
Sensible heat in dry fuel gas, B.t.u.	9.128	9.128	9.128	22.449	22.449	22.449	99.61	99.61	99.61
Sensible and latent heat in moisture, B.t.u.	16.682	16.682	16.682	94.420	94.420	94.420	81.27	81.27	81.27
Latent heat of combustion, 1 lb. dry fuel gas, B.t.u.	1,253.800	1,253.800	1,253.800	1,253.800	1,253.800	1,253.800	1,253.800	1,253.800	1,253.800
Total, B.t.u.	1,279.610	1,279.610	1,279.610	1,380.669	1,380.669	1,380.669	1,434.68	1,434.68	1,434.68
COMBUSTION AIR.									
Sensible heat in dry air, B.t.u.	6.292	6.292	6.292	6.292	6.292	6.292	6.292	6.292	6.292
Sensible and latent heat in moisture, B.t.u.	7.729	7.729	7.729	7.729	7.729	7.729	7.729	7.729	7.729
Total, B.t.u.	14.021	14.021	14.021	14.021	14.021	14.021	14.021	14.021	14.021
Total heat expended, B.t.u.	1,293.630	1,293.630	1,293.630	1,394.69	1,394.69	1,394.69	1,448.70	1,448.70	1,448.70
TOTAL HEAT IN PRODUCTS OF COMBUSTION ABOVE 32° F.									
Sensible heat in dry products of combustion, B.t.u.	144.65	183.14	228.69	144.65	183.14	228.69	144.65	183.14	228.65
Sensible and latent heat in moisture, B.t.u.	31.41	33.89	36.37	117.24	123.12	129.24	94.16	99.07	104.08
Total heat in products of combustion, B.t.u.	176.06	217.03	265.06	261.89	306.26	357.93	238.81	282.21	332.77
HEAT BALANCE.									
Total heat expended, B.t.u.	1,293.630	1,293.630	1,293.630	1,394.69	1,394.69	1,394.69	1,448.70	1,448.70	1,448.70
Total heat lost in products of combustion, B.t.u.	176.06	217.03	265.06	261.89	306.26	357.93	238.81	282.21	332.77
Total heat available per lb. dry fuel gas consumed, B.t.u.	1,117.57	1,076.60	1,028.57	1,132.80	1,088.43	1,036.76	1,209.89	1,166.49	1,115.93
Total heat available per lb. dry fuel gas consumed, per cent.	86.39	83.23	79.51	81.22	78.04	74.33	83.52	80.31	77.03
Total heat dissipated in products of combustion, per cent.	13.61	16.77	20.49	18.78	21.96	25.67	16.48	19.69	22.93

washed gas under furnace-top conditions carrying 35 grains moisture. The lowest percentage of available heat is shown to exist for washed gas cooled to 125° F. Under conditions of this table it may be interesting to note that thermal equilibrium, raw unwashed gas against washed gas, approximates a temperature midway of 70° and 125° F.

Solids

The solids contained in the gas consist of, first, the mechanically carried particles, of varying degrees of fineness and, second, a volatile substance which is carried as a gas and finally deposited or which may pass through the entire system.

Mechanically Carried Flue Dirt.—The table following shows the analysis of flue dust, using 90 per cent. Mesabi ores, for the year 1912:

Analysis of Downcomer Dust for Year 1912

Month	SiO_2	Fe	Mn	Phos.	H_2O.	Al_2O_3	Bases	Unflux
January	7.30	44.03	0.74	0.060	14.50	2.34	1.90	7.74
February	8.26	40.58	0.69	0.070	14.20	2.69	2.38	8.57
March	8.18	38.48	0.80	0.068	21.30	2.77	1.64	9.31
April	10.36	36.29	0.61	0.066	15.60	2.67	2.47	10.56
May	11.90	37.71	0.68	0.082	14.30	2.78	2.78	12.61
June	9.66	40.88	0.65	0.071	10.10	2.74	2.38	10.02
July	8.89	38.37	0.58	0.102	10.60	2.57	3.54	7.92
August	8.01	35.01	0.68	0.101	17.50	2.46	2.73	7.74
September	8.66	40.28	0.70	0.118	12.20	2.86	2.85	8.67
October	9.34	40.29	0.56	0.093	13.20	2.58	2.25	9.67
November	6.27	38.95	0.52	0.054	14.40	2.17	1.51	6.93
December	8.77	40.58	0.50	0.071	14.90	2.66	2.17	9.26
Average	8.80	39.29	0.64	0.080	14.48	2.61	2.38	9.03

Chemical and Physical Characteristics of Flue Dust.—The gas from the furnace should deposit the heavier material of the flue dust in the dust catcher and pockets, and only the lighter material should pass to the scrubbers. In order to determine if this is true, we took samples of dust from the dust catcher and also samples of the sludges from the settling basins, to ascertain the qualities—both physical and chemical. We first endeavored to sieve the material, after drying, but with no success, and then washed it through screens, weighing that which remained. The screens ran up to 300-mesh, and the tests show that 92 to 95 per cent. of the sludge from the basin passed through the 300-mesh screen. This material is also the richest in metallic yield, showing 41 per cent. iron. It is noticeable that only a low percentage of fine dust, that is through a 300-mesh screen, remained in the dust catcher—18 per cent. in

one case and 5 per cent. in another. This may have been carried down in contact with larger particles; that is, the fine particles would stick on the larger ones, but would be separated by water. I would mention that very interesting data would be revealed if ore were wet sieved instead of dry sieved: a more uniform and more accurate representation of the actual physical condition would then be had. An ore which we sieved dry, ran 9 per cent. through 100-mesh, and the same sample ran 24 per cent. through 100-mesh when wet sieved.

Wet Sieve Analysis

Dust direct from No. 6 dust catcher			Dust direct from scrubber	
Weight grams	Per cent. weight	Remaining on	Weight grams	Per cent. weight
0.60	0.79	20		
2.80	3.71	40	0.17	0.26
3.10	4.10	60	0.20	0.31
6.55	8.67	80	0.29	0.45
2.90	3.84	100	0.20	0.31
19.60	25.94	150	0.81	1.25
18.60	24.62	200	1.63	2.52
7.60	10.06	300	1.40	2.16
13.80	18.27	Through 300	60.00	92.74
Total....75.55			64.70	

Wet Sieve Analysis

Dust direct from scrubber			From No. 2 furnace dust catcher	
Weight grams	Per cent. weight	Remaining on	Weight grams	Per cent. weight
		20	18.8	2.62
0.05	0.08	40	165.8	23.13
0.25	0.40	60	113.9	15.89
0.15	0.24	80	175.3	24.45
0.15	0.24	100	31.8	4.43
0.32	0.51	150	138.2	19.26
0.70	1.12	200	34.8	4.85
1.45	2.33	300	.....	.....
59.30	95.08	Through 300	.....	.....
		Through 200	38.2	5.35
Total....62.37				

Analysis of Flue Dust Direct from Scrubber Water. Wet Sieving

	On 150	On 200	On 300	Through 300
Silica	10.85	14.02	13.85	15.00
Iron	20.23	26.11	28.89	41.11
Manganese	.....	.....	.....	0.72
Alumina	.....	.....	.....	7.64
Lime	.....	.....	.....	1.55
Magnesia	.....	.....	.....	0.15
Ignition loss	53.42	39.91	34.55	15.92

Analysis of Settling from Settling Basin

	No. 1 Bay	No. 5 Bay
Silica	13.53	11.92
Iron	43.00	45.75
Alumina	8.00	6.09
Ignition loss	11.35	10.00

Settling Properties of Flue Dust.—Bearing in mind the finely divided condition of the dust, as shown in the sieve tests, it will be interesting to note the rate of settling of this material. We have a double settling basin. The water passes from the collecting sewer beneath the scrubbers and flows into one section, filling it with dirt. The dirty water is then diverted to the other section, and the first cleaned by means of a grab bucket, the material being loaded into cars. There are six partitions, or baffles, and wooden gates which tend to hold back the dirt and aid in settling it. Each division of the basin is 161 ft. long and 26 ft. wide, the water flowing over a weir at the end. The capacity is 500 tons of wet dirt. Each side of the settling basin holds 197,920 gal. of water when clear of dirt. The dirty water was allowed to flow through for a day; then, from a platform erected over the first compartment, samples were taken at various depths. A brass tube, fitted with a rubber stopper and a rod, was used to obtain the samples. The closed tube was lowered to the desired depth, the bottom opened and the water allowed to flow in. The stopper was then drawn into place and the sample put in a jar for weighing and measuring. The results are shown in Table III.

Time Settling Tests With Flue Dust and Sludge from Gas-Cleaning Plant.—The sludge in the settling basin is of the consistency of a thick jelly and filters with almost the same rapidity. In this condition it is extremely difficult to handle. We allow the basin to drain as much as possible, then remove the sludge with a grab bucket, and load into cars. These cars must be calked water tight to hold the material. If the material would drain easily most of the trouble would be obviated, and tests were made to locate the difficulty. A flume was first constructed to carry the water from one scrubber and with sufficient fall to create a

TABLE III.—*Amount of Dust at Various Levels and for Different Periods of Settling*

Unsettled Solution				Series A (After settling 18 min.)			
Level Feet	Time	Volume	Grains Dust per Gal. Water	Level Feet	Time	Volume	Grains Dust per Gal. Water
1	A. M.	200 c.c.	37.00	1	3:14	193 c.c.	5.78
2	2:39	229 "	32.00	2	3:17	201 "	4.15
3	2:42	209 "	33.00	3	3:20	185 "	3.97
4	2:45	245 "	37.50	4	3:23	142 "	4.10
5	2:48	154 "	36.50	5	3:26	172 "	4.15
6	2:51	191 "	73.60				

Series B (After settling 36 min.)				Series C (After settling 51 min.)			
Level Feet	Time	Volume	Grains Dust per Gal. Water	Level Feet	Time	Volume	Grains Dust per Gal. Water
1	3:32	188 c.c.	9.46	1	3:47	206 c.c.	3.77
2	3:35	185 "	5.38	2	3:50	181 "	3.20
3	3:38	195 "	3.62	3	3:53	203 "	3.38
4	3:41	233 "	2.26	4	3:56	128 "	2.80
5	3:44	175 "	3.90	5	3:59	171 "	3.22

Series D (After settling 66 min.)				Series E (After settling 76 min.)			
Level Feet	Time	Volume	Grains Dust per Gal. Water	Level Feet	Time	Volume	Grains Dust per Gal. Water
1	4:02	212 c.c.	4.90	1	4:17	221 c.c.	3.45
2	4:05	197 "	3.62	2	4:20	220 "	4.22
3	4:08	193 "	3.85	3	4:23	202 "	4.80
4	4:11	134 "	5.84	4	4:26	213 "	5.15
5	4:14	160 "	3.68	5	4:29	165 "	3.32

fairly rapid flow. Constrictions were built in the flume so as to speed up at different points to carry the lighter material forward. By this scheme we succeeded in making a rough separation, and obtained a fine but granular product from which the water drained easily. The trough was widened out somewhat at the end and the speed of flow decreased. This gave us the same product we had to deal with in the basins, as a lot of fine material was deposited. In order to determine a settling standard for comparison, we took some Group-3 ore, a standard Mesabi, and wet sieved it through the various 60–80–100–150–200–300 mesh screens. The 300-mesh screen was the finest we could get. These different screenings were put into similar bottles filled with water and shaken up. Following is the time of settling of the different sizings, fixed by numerous observations:

Screen	Time of Settling, Seconds
60	$1\frac{3}{4}$
80	3
100	4
150	$7\frac{1}{2}$
200	14

The material which passed through 300-mesh required several hours to settle clear. This was also divided by time intervals; that is, the material was shaken and allowed to settle for 2 min., and the water and suspended material poured off. This was repeated until a standard, clearing itself in 2 min., was prepared. Thus different time separations were made. Of course we could not determine the fineness of this material, but could only recognize it as that which settled in a certain interval of time. The sludge obeyed the same rules and that which came down in less than 5 min. filtered with comparative ease. A scum of extremely fine, light material, principally coke dust, extremely gelatinous, forms on top of the basins and afterward sinks. A small quantity of this material, if mixed with the coarse dirt which previously filtered freely, at once prevents all filtration. It is reasonable to suppose that a lot of the dirt could be easily handled if this slime was eliminated. We are at present experimenting with various means to make this separation and expect to do away with many of the difficulties in handling this material and also recover a good product for sintering or charging back. An analysis of the material floating on the top is exhibited:

	Dry	Wet
Silica	22.54	
Iron	24.40	
Manganese	0.61	
Phosphorus	0.068	
Moisture		57.00
Alumina	7.43	
Lime	1.52	
Magnesia	0.25	
Ignition loss	40.84	

Quantity of Dust Carried by Furnace Gas.—An attempt was made to arrive at a correct figure for dirt in the gas entering the scrubbers, and it was decided that 500-c.c. samples of water should be taken at intervals from the overflow water of the scrubber and collected in a carboy. The water was afterward filtered. In this manner and over a period of 19 days, samples were taken, consisting of 2,000 lb. of water, or 240 gal. From our weir readings on the settling-basin overflow, we determined the quantity of water for the period, and by means of recording meters and Pitot-tube readings we determined the gas volume passing through the scrubbers, per gallon of water, and therefore determined the actual dirt represented in 6,240 cu. ft. of gas, which was 2.6 grains per cubic foot. The sampling was done under normal operating conditions. The results shown in the following table are as nearly accurate as it is possible to reach under the conditions at this plant.

Computation of Dust Carried by Furnace Gas for Month of June, 1913

Gas cleaned, cu. ft.	3,853,044,000
Water consumed, gal.	152,762,000
Average dust content of scrubber water per U. S. gal., grains	60.345
Gas cleaned per hour, $\frac{3,853,044,000,}{720}$ cu. ft.	5,351,450
Water consumed per hour, $\frac{152,762,000,}{720}$ gal.	212,169
Grains dust removed per cu. ft. gas, $\frac{212,169 \times 60.345,}{5,351,450}$ grains	2.393
Average dust content of primary gas per cu. ft., grains	0.20
Grains dust per cu. ft. raw gas (2.392 plus 0.20)	2.592

Note.—Gas consumed at standard conditions of 62° and 30 in. Hg.

Blast-furnace gas at the scrubbers, from the foregoing determinations, contains 2.592, say 2.6 grains of dust per cubic foot. This dust consists of ore, coke dust, and limestone dust, beside organic acids and compounds which seem to condense and are composed principally of silica and alumina. The gas may be of a dense white nature, or of a reddish hue, due to the predominance of fume in the former case, and ore dust in the latter case. The former is the harder to remove, and almost invariably if the dirt on the filter capsule in the testing apparatus is light in color, the gas is dirtier than if red. Gas after final cleaning still retains some of the whitish fume. This material is deposited in the bottom of the stoves, in the stacks, and on the boiler settings. In places where the gas has only been rough cleaned the color is a pinkish white, without grit and with a feeling similar to talcum powder. The stacks at all times show this white fume, and a determination was made to see how much of the dust left in the gas at the scrubbers passed through the entire system without depositing. Below is that determination and an analysis of the material that was deposited in the stoves and boilers. It is seen that 27.5 per cent. of the dirt shown in the scrubber analysis passes through the entire system.

Wet Sieve Test on Deposit Found in No. 2 Stove, No. 6 Furnace

Remaining On	Weight, Grams	Percentage
10	1.10	1.77
20	0.54	0.87
40	0.54	0.87
60	0.29	0.46
80	0.51	0.82
100	0.62	1.00
150	0.80	1.29
200	0.75	1.24
Through 200	56.85	91.69
	Total, 62.00	

Nature and Quantity of Dust Passing through System and Exhausted to Atmosphere.—Material that passed through the washing plant with the gas and found in the flue gas of No. 15 boiler, No. 3 boiler house extension —clean gas:

Silica		22.80
Iron	(=Fe_2O_3 8.57 per cent.)	6.00
Manganese	(=Mn_3O_4 2.09 per cent.)	1.50
Alumina		15.73
Lime		21.42
Magnesia		2.50
Potassium oxide		20.90
Ignition loss		6.08

Deposit Found on the Vanes of the Theisen Washers

Silica		23.84
Iron	(=Fe_2O_3 9.43 per cent.)	6.50
Manganese	(=Mn_3O_4 3.52 per cent.)	2.58
Alumina		12.65
Lime		15.52
Magnesia		1.62
Potassium oxide		6.96
Ignition loss		23.90

The dust in the gas at the top of No. 3 fan separator was 0.09171 grain per cubic foot; the dust in exit gases and boiler (referred to fuel gas), 0.0252 grain per cubic foot. The amount passing through the system equaled 27.50 per cent. This test was taken when the gas volume was low, which accounts for the low dust content.

The material passing through the final cleaners and deposited on the piston heads and at the several outlet ports of the engine, analyzed as follows:

	Per cent.
SiO_2	18.40
Mn_3O_4	2.40
Al_2O_3	15.55
CaO	12.20
MgO	1.68
Fe_2O_3	4.71
Alkalies	42.89
Ignition loss	2.17

Description of Plant

The system of cleaning gas at the Duquesne blast furnaces consists at present of scrubbers, fans, and Theisens. This installation was first

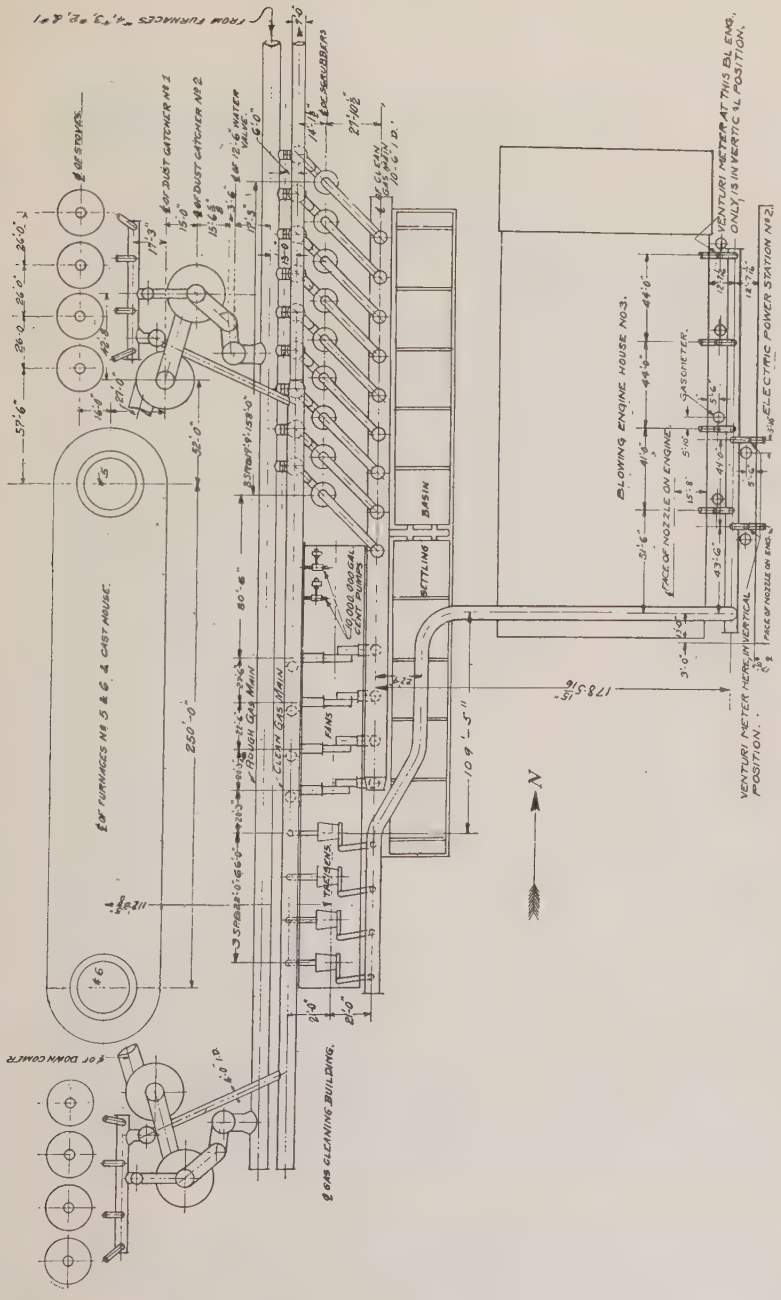

FIG. 1.—GENERAL ARRANGEMENT OF THE PLANT.

designed for the purpose of cleaning gas for six 3,600-h.p. gas engines, four sets of stoves, and 1,500 boiler horse power. At the time of its installation it was the largest in the country and very little reliable data

FIG. 2.—DUQUESNE GAS-CLEANING PLANT, SHOWING SCRUBBERS, WITH INTAKE FROM DIRTY-GAS MAIN AND OFFTAKE TO COLLECTING MAIN; ALSO SETTLING BASINS.

could then be collected on the subject. Nine towers, four fans, and four Theisens were erected. After operating a while, numerous changes became necessary. We found that four scrubbers, two fans, and two Theisens were sufficient for the demands. The plant is shown in the

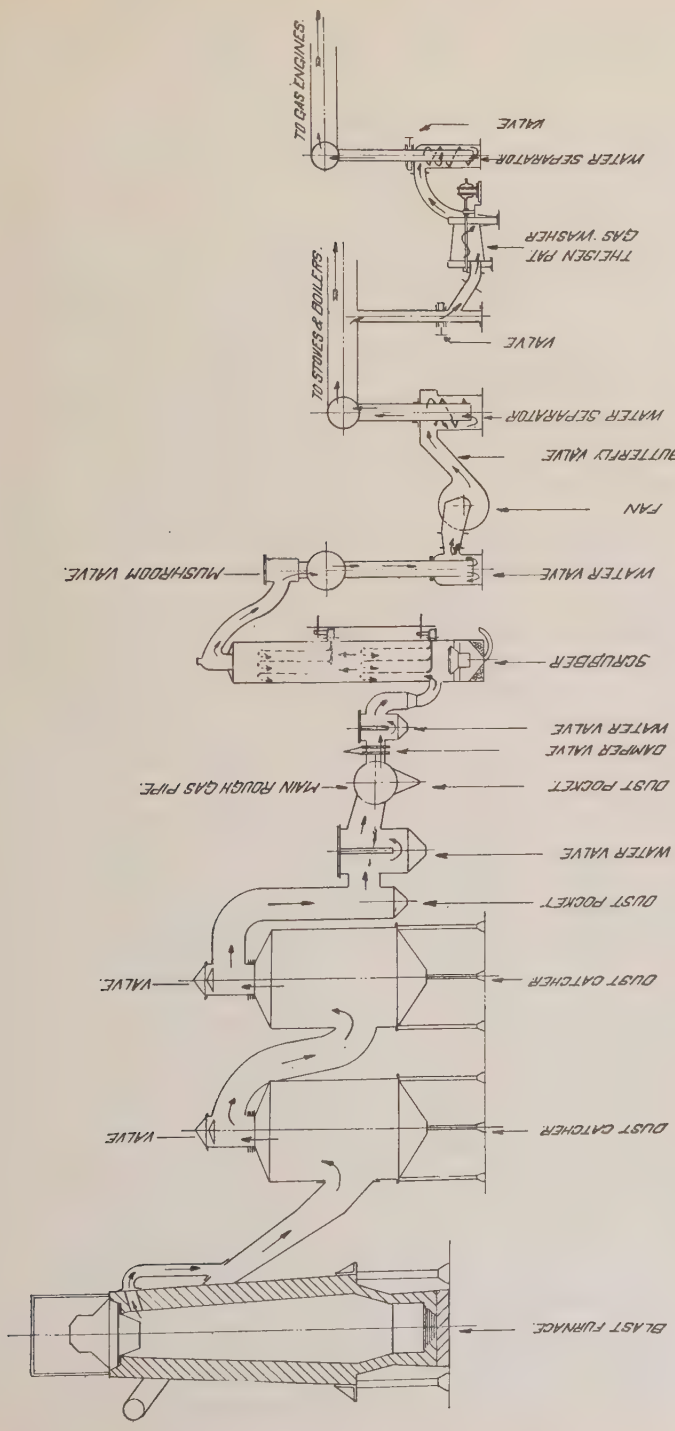

FIG. 3.—THE COURSE OF THE GAS THROUGH THE CLEANING PROCESSES TO STOVES, BOILERS, AND ENGINES.

illustrations as at present. Fig. 1 is a general plan and Fig. 2 a view of the plant. Fig. 3 shows the course of the gas through the cleaning processes.

Scrubbers

We first started with scrubbers having a rotating nozzle on the top, the water to form a rain and thus come in contact with the gas and clean, as well as cool it in this manner. The gas channeled under these conditions, the pressure over the scrubber area being constant; but the ascending gas column, due to velocity head, deflected the water. This was shown clearly by repeated tests, using a pipe with a funnel on one end and a standard pail outside, the time of the filling of the bucket being taken. After the bucket was full, the pipe was withdrawn 4 in. and the filling timed once more. In this way we tested across the scrubber. Distinct dry areas were found, irrespective of all possible regulations of water and deflecting gas schemes. It was then decided to reverse the scheme of having gas the dominating element and put the water in in such manner as to change the direction of the gas in its upward course and at the same time spray it repeatedly. This was done by injecting the water through a valve having a number of openings and a core moving therein. This core successively blocks off the openings to the different nozzles, and thereby temporarily stops the flow of water, causing an area of low pressure directly above the nozzle. When the core has passed, the water resumes and sprays through the gas which has flowed to this point, driving it to the next nozzle, where the same process is continued. The core is revolved electrically, a 5-h.p. motor being ample for four valves—*i.e.*, two scrubbers, or 60,000 ft. of gas. The gas rises through the scrubber at the rate of 4 ft. per second and the water at 60 ft. per second with a head of 35 lb. main pressure. A second valve is placed in the scrubber just above the range of the lower one and operates in the same manner. Screens are placed above the nozzles to break up the water into fine drops so as to bring the gas and water into intimate contact. It is evident that the gas will be repeatedly sprayed as it travels in an upward spiral course until it has passed the range of the water. Fifteen revolutions per minute is found to be about the right speed for the core. Fig. 4 is a section through the scrubbers.

The gas is cooled down very effectually, numerous readings showing only 5° to 6° above the temperature of the ingoing water. An average moisture test shows 0.5 grain above the saturation point at the temperature of the outgoing gas. These tests were taken on the outlet pipe of the scrubbers and were made with the Brady apparatus and also with the Brown apparatus. The difference in temperature between the gas and water can be increased or decreased, depending on the quantity of gas

passing through the scrubbers. We have found that between 25,000 and 30,000 cu. ft. can be passed through our 12-ft. scrubbers, and although the average difference in temperature is higher than that above mentioned, for the year 1912, yet it is the most convenient working point from a financial standpoint, if not so efficient thermally. In other words, the

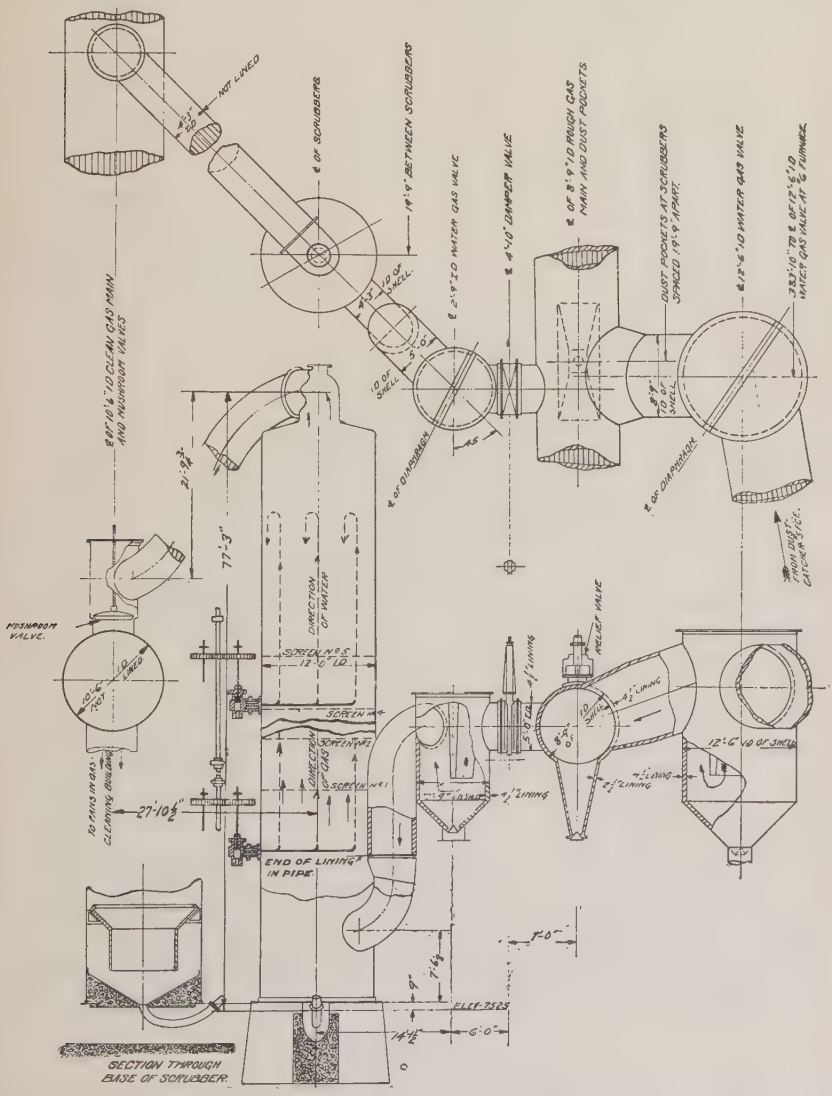
NOT LINED
℄ OF SCRUBBERS
14'-9" BETWEEN SCRUBBERS
℄ 2'-9" I.D. WATER GAS VALVE
℄ 4'-0" DAMPER VALVE
℄ OF 8'-9" I.D. ROUGH GAS MAIN AND DUST POCKETS
DUST POCKETS AT SCRUBBERS SPACED 19'-9" APART.
℄ 2'-6" I.D. WATER GAS VALVE
33'-10" TO ℄ OF 12'-6" I.D. WATER GAS VALVE AT ℄ FURNACE
℄ OF DIAPHRAGM
MAIN DUST CATCHER STACK
MUSHROOM VALVE.
DIRECTION OF WATER
SCREEN No 3
SCREEN No 2
SCREEN No 1
DIRECTION OF GAS
27'-10½"
END OF LINING IN PIPE
½" LINING
12'-6" I.D. OF SHELL
SECTION THROUGH BASE OF SCRUBBER.

FIG. 4.—SECTION THROUGH THE SCRUBBERS SHOWING THE METHOD AND APPARATUS FOR PRIMARY CLEANING. THE CUT-OFF VALVES ARE ON THE SIDE AND DRIVEN FROM A VERTICAL SHAFT.

moisture content of the gas increases very rapidly above 90° F. and we aim to keep under this as much as possible. The moisture, if not controlled within limits, will lower the stove efficiency to such an extent as to make cleaning prohibitive.

The water is taken from the service mains and is charged at the regular water rate. This condition is different in plants where waste water is used. The Duquesne plant construction is sucli as to deliver our bosh cooling water to mill condensers and water-purification plants. About 815 gal. of water are used per minute per scrubber. All water flows over a weir, so that the quantities are accurately determined, readings

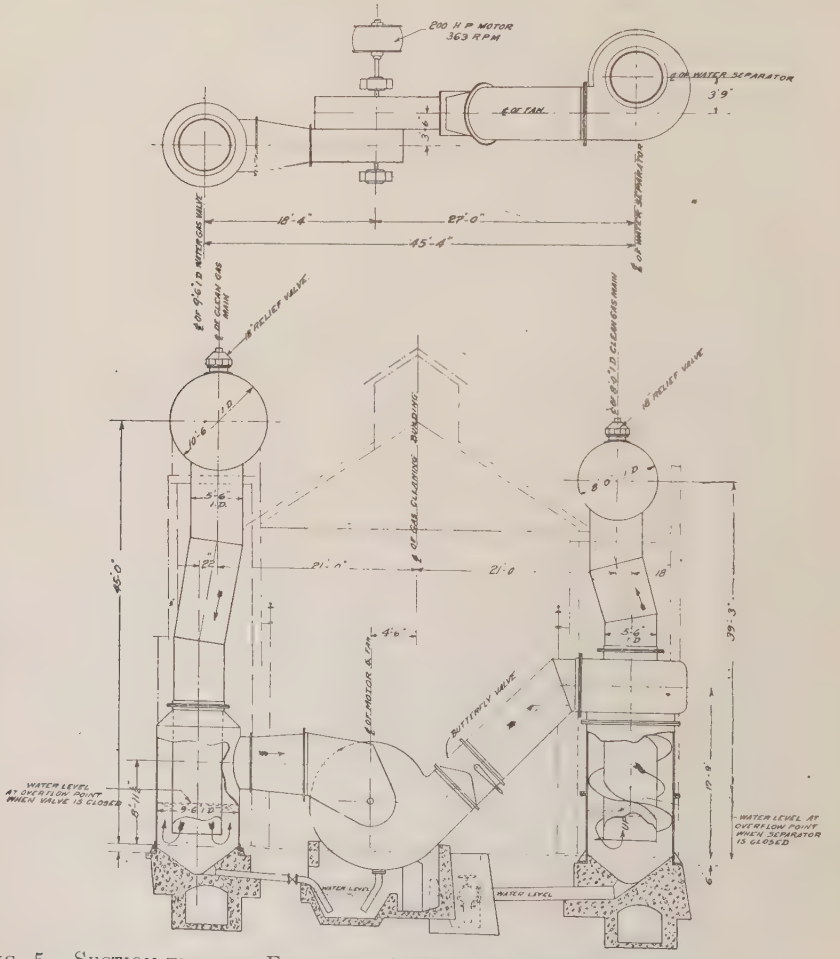

Fig. 5.—Section through Fans, and Method of Passing Gas through Seals and Separators.

being taken four times a day. The electric power is metered, and the quantity of gas to the engines is determined by Venturi recording instruments. The stove gas is determined by Pitot-tube readings, hydro determinator, and Ellison gauge, all results checking closely.

The water used in gas valves and seals is that which is used to keep the

gas from entering the scrubbers not in use and a small stream flows continuously to preserve these seals. All water flows to the settling basins and is therefore recorded in the weir readings.

Fig. 6.—Water Seal Between Collecting Gas Main from Scrubber and the Four Fans.

Fans

The fans, Fig. 5, are of Sturtevant type, electrically driven by direct-current and alternating-current motors, and each has a capacity of 84,000 cu. ft. of gas per minute. A water seal, Fig. 6, is placed on either side to provide means for repairing. The seal after the fan is also

a spiral separator for water which may be picked up in the fan. A calked main serves to connect the scrubbers with the fans and is 10 ft.

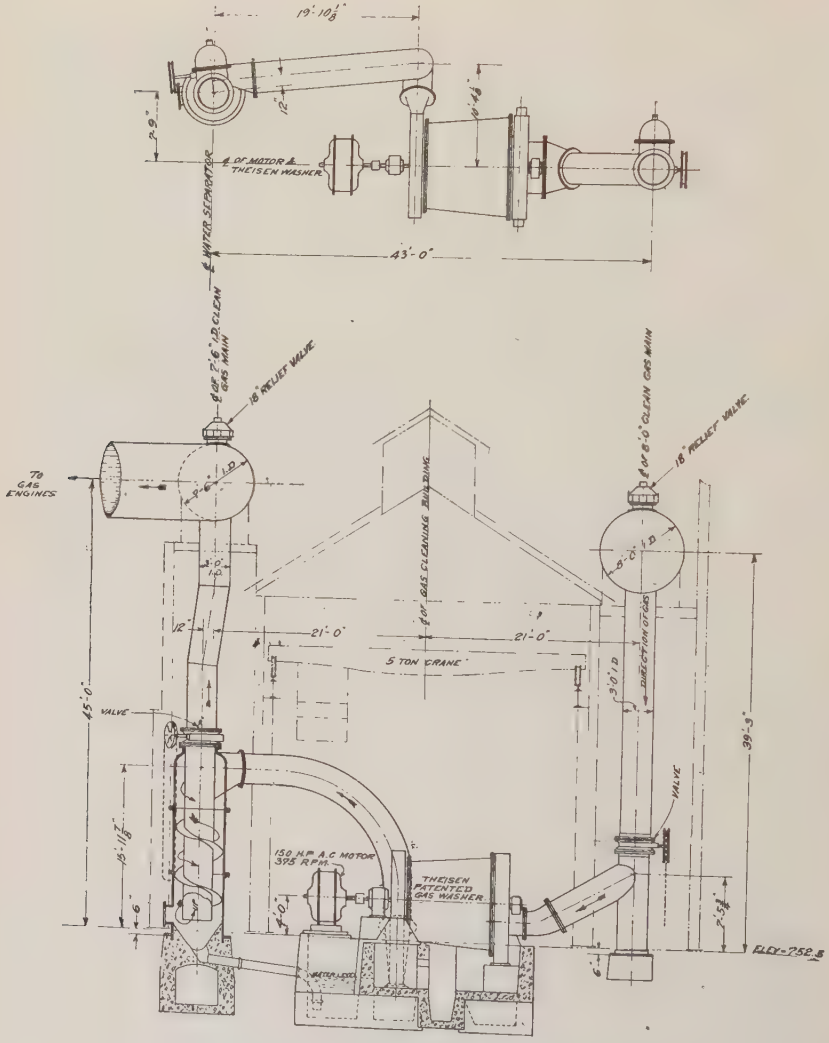

Fig. 7.—Section through Theisens, Seals, and Separators.

6 in. in diameter. After leaving the fans the gas is carried into an 8-ft. main, or primary gas main, and distributed.

Theisens

The Theisens, Fig. 7, are four in number, three driven by induced and one by direct-current motors. The motors are each 150 h.p. and drive

the rotor 375 rev. per minute. Each Theisen has a capacity of 15,000 cu. ft. of gas per minute. There are seals and separators on the Theisens the same as on the fans.

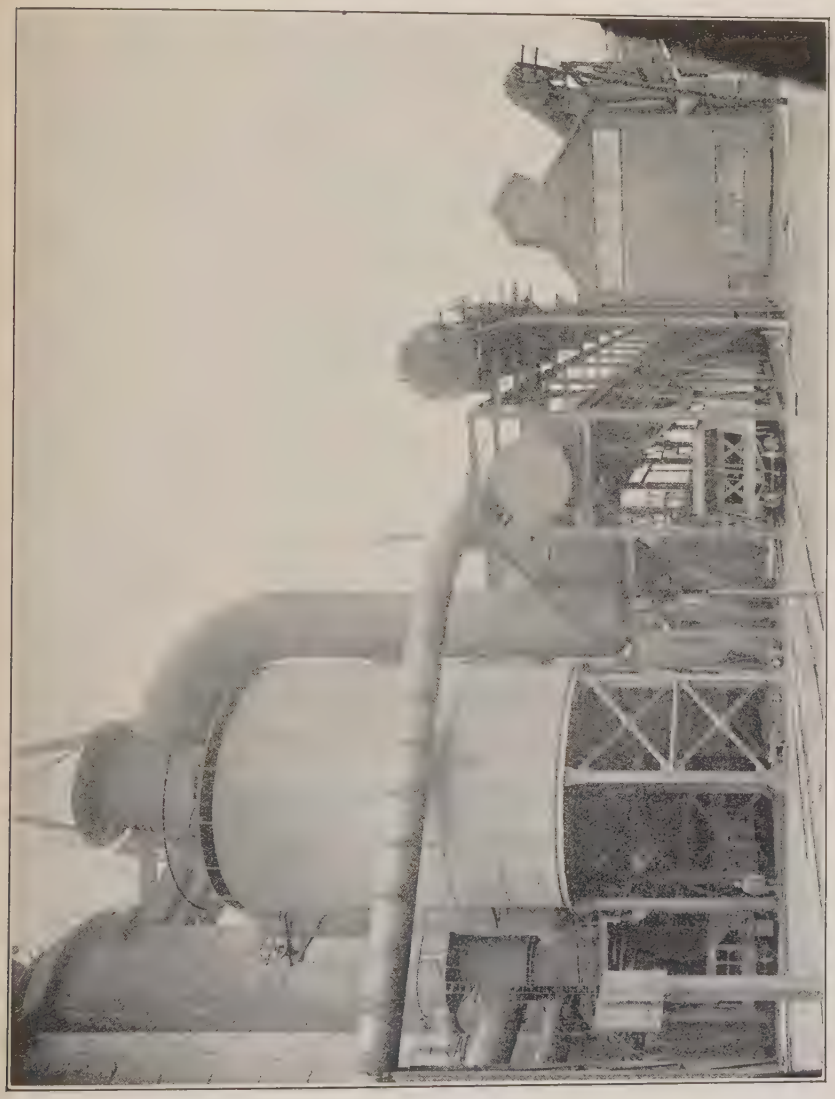

FIG. 8.—DIRTY-GAS MAIN WITH CLEAN-GAS MAIN AT HIGHER LEVEL, ALSO THE END OF THE REFINED-GAS MAIN LEADING FROM THE THEISENS TO THE ENGINES

Settling Basins

The settling basins are 161 ft. in length, 26 ft. wide, and 11 ft. 6 in. total depth. The settling depth is 6 ft. There are two sections and these are used alternately. The sewer from the scrubbers into which the dirt and water is collected is so situated as to make possible the diversion

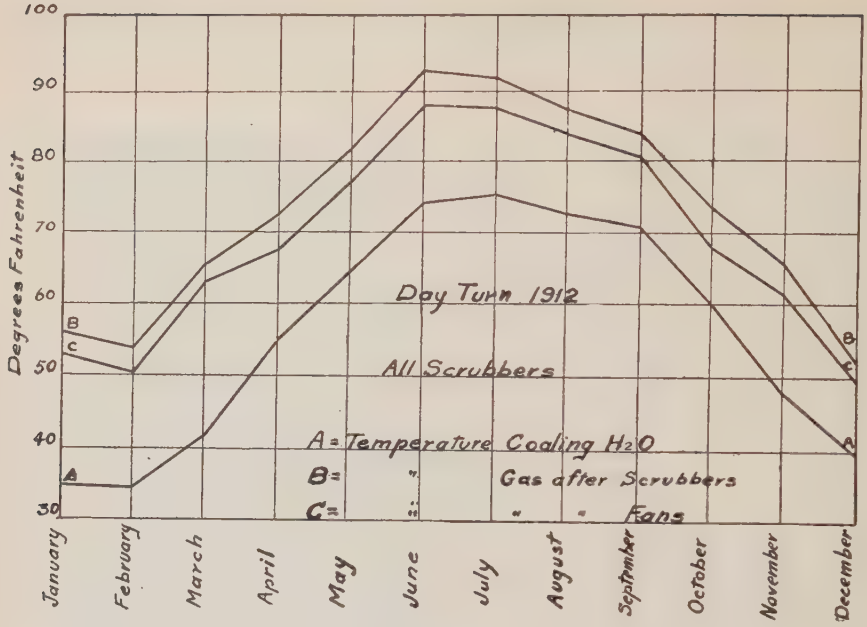
Degrees Fahrenheit
100
90
80
70
60
50
40
30
Day Turn 1912
All Scrubbers
A = Temperature Coaling H_2O
B = " Gas after Scrubbers
C = " " Fans
A
B
C
January
February
March
April
May
June
July
August
September
October
November
December

Fig. 9.

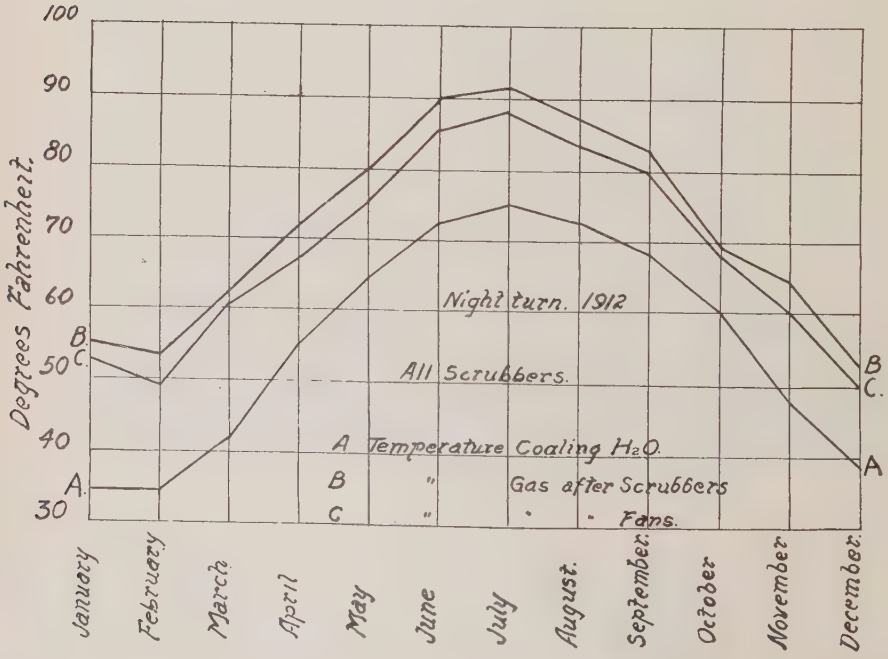
Degrees Fahrenheit.
100
90
80
70
60
50
40
30
Night turn. 1912
All Scrubbers.
A Temperature Coaling H_2O.
B " Gas after Scrubbers
C " " Fans.
A.
B
C.
A
B
C.
January
February
March
April
May
June
July
August.
September.
October
November
December.

Fig. 10.

of the water to either compartment. When the basins are filled with dirt the water is drained and the material loaded into cars by a grab bucket. Partitions are raised in the basins to aid in the retardation of the dirt.

Operation of Plant

The gas from six blast furnaces is allowed to go into a general dirty-gas main, 8 ft. 6 in. in diameter, shown in Fig. 8, and from here is distributed

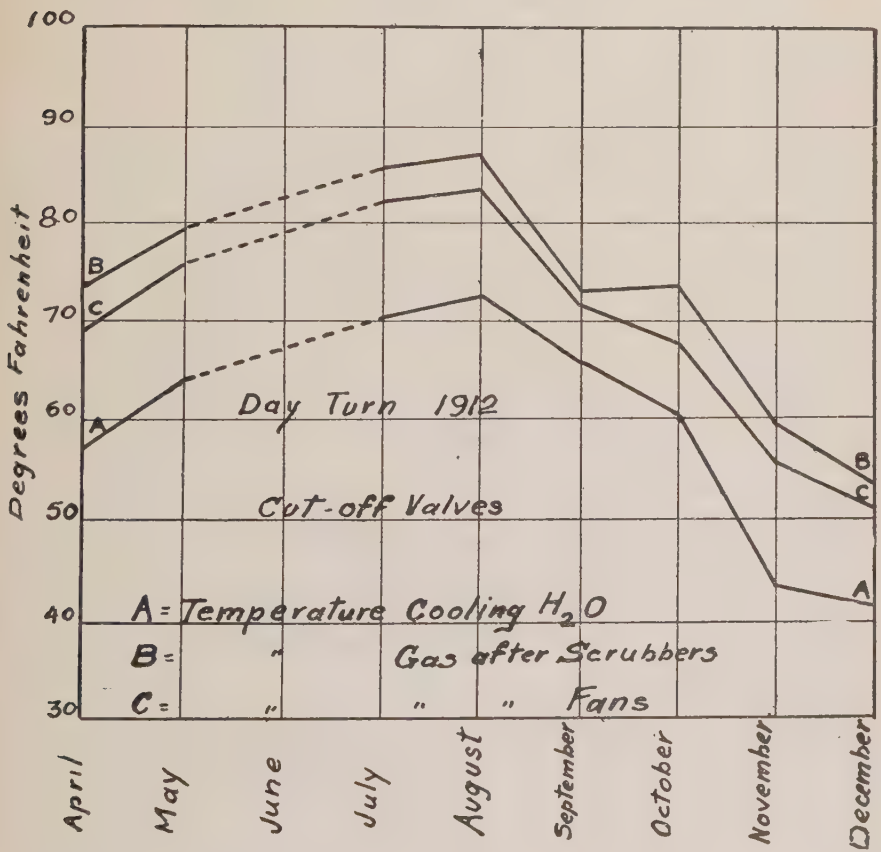

Fig. 11.

to various destinations, as follows: No. 1, 2, 3, and 4 boiler houses, No. 1 and 2 sets of stoves, and gas-cleaning plant. A pressure of 12 in. of water is maintained in this main. The gas to the gas-cleaning plant goes through water valves to scrubbers, in parallel, going only through one and thence to the 10 ft. 6 in. collecting main, from which it goes through the fans to the primary or scrubbed-gas main and is distributed

to No. 3 boiler house extension, stoves on No. 3, 4, 5, and 6 furnaces, and to the Theisens.

The pressure in the receiving main is $10\frac{1}{2}$ in. and in the primary main 8 in. The Theisens boost the pressure to 12 in. and deliver to the engines. These pressures relate to operating without fans, the gas being propelled by the pressure of the furnace. We operated for a period of a

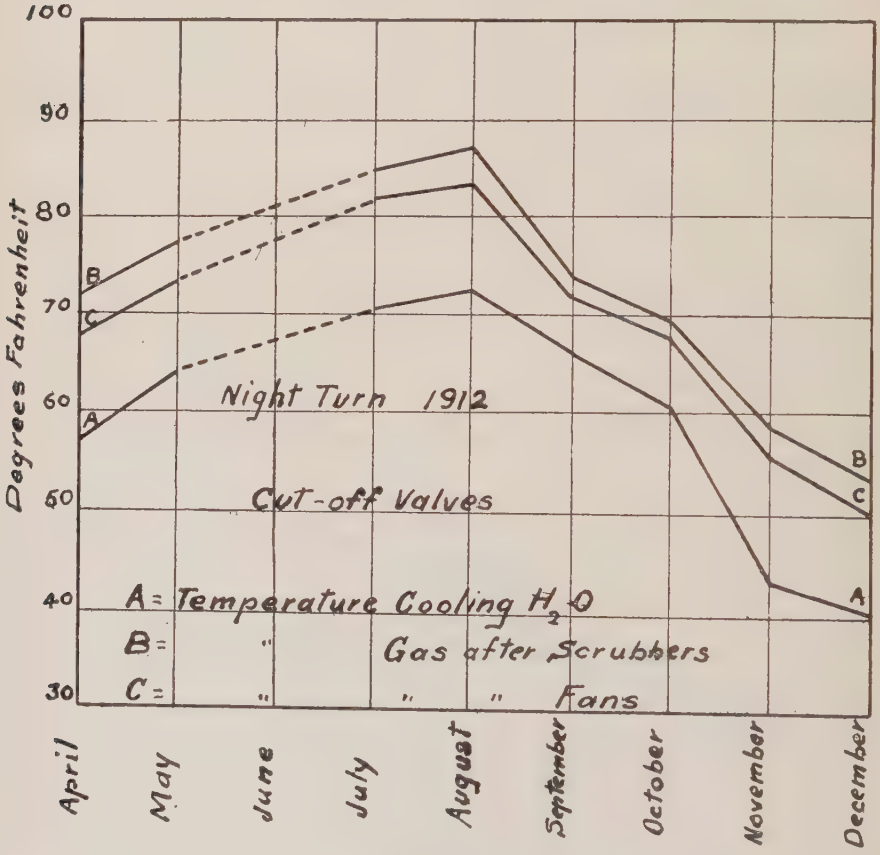

FIG. 12.

year and a half this way, but are using fans now, due to a desire to lower the top pressure, saving gas losses, and lessening the danger in operation during tuyère changing, etc., when a high gas-main pressure causes trouble. We desire a pressure of 5 to 6 in. at the stove burners, for the reason that our distributing mains are too small and we have poor stove draft. The friction through the various fan seals, and through the fans also, made it necessary to back up the gas. In a new installation many of these difficulties would be obviated. Formerly we removed no dirt

in the fans, but we have so altered them lately as to reduce the dirt content from 0.2 to 0.05 and 0.06 grain per cubic foot.

The Theisens, as shown in the gas test for 1912, lower the dirt content to 0.009 grain per cubic foot.

Data of our operating conditions for 1912 are shown in Tables IV to VII and Figs. 9 to 12.

TABLE IV.—*Gas Cleaned and Water Consumed, Primary Plant, Year* 1912

Month	Gas Cleaned	Water Consumed			Cu. ft. Gas Cleaned per Gal. Water Consumed		
		Scrubbers	Water Gas Valves	Total	Scrubbers	Water Gas Valves	Total
Jan.....	4,180,005,000	128,464,000	13,392,000	141,856,000	32.5	304.7	29.5
Feb.....	3,718,755,000	123,424,000	16,096,000	139,520,000	30.1	231.0	26.7
Mar.....	4,044,490,000	128,161,000	13,392,000	141,533,000	31.6	302.0	28.6
Apr.....	3,947,919,000	123,122,000	13,140,000	136,262,000	32.1	300.5	29.0
May.....	3,906,599,000	124,947,000	13,392,000	138,339,000	31.3	291.7	28.2
June.....	3,747,409,000	123,941,000	12,960,000	136,901,000	30.2	289.2	27.3
July.....	3,736,670,000	127,090,000	13,392,000	140,482,000	29.4	279.0	26.6
Aug.....	3,790,763,000	127,760,000	13,392,000	141,152,000	29.7	283.0	26.9
Sept.....	3,885,908,000	146,491,000	12,960,000	159,451,000	26.5	299.8	24.4
Oct.....	4,488,127,000	155,525,000	13,492,000	169,017,000	28.9	332.7	26.6
Nov.....	3,922,370,000	149,861,000	12,960,000	162,621,000	26.2	302.7	24.1
Dec.....	4,141,773,000	151,625,000	17,856,000	169,481,000	27.3	231.9	24.4
				Average	29.7	287.4	26.9

The water shown as used in water-gas valves is only used as a seal and does no washing, so that the figure for scrubbers is the proper one to consider.

TABLE V.—*Recapitulation of Operators' Daily Report, Year* 1912

No. 3, 4, 5, and 6 Scrubbers, Cut-off Type

DAY TURN.

Month.	Temperature Cooling Water.			Temperature Gas after Scrubbers.			Temperature Gas after Fans.			Temperature Gas after Theisens.			Temperature Atmosphere.		
	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.
Apr.	61.0	55.0	57.4	81.0	69.0	73.7	75.0	66.5	69.2	68.0	60.5	64.2	71.5	45.0	59.9
May.	68.0	59.5	64.1	84.5	73.5	79.3	81.5	70.0	75.8	75.0	65.5	70.7	82.5	64.5	75.2
July.	73.5	69.0	70.6	59.5	83.0	85.6	85.0	79.5	82.2	78.0	75.0	75.4	77.0	66.0	72.5
Aug.	76.0	67.5	73.0	93.0	83.5	87.1	89.0	78.5	83.4	81.5	73.0	78.1	81.5	64.5	73.7
Sept.	70.0	60.0	66.2	77.0	67.5	73.3	76.0	65.0	71.7	74.5	65.0	70.8	73.0	52.0	63.5
Oct.	67.0	52.0	60.6	77.5	62.5	73.6	76.0	60.0	68.0	74.0	57.5	55.9	75.5	47.0	60.3
Nov.	46.0	40.0	43.2	70.5	53.0	59.8	65.5	48.0	55.9	54.0	46.0	50.3	54.0	35.0	43.0
Dec.	47.0	37.0	41.1	67.0	43.0	53.7	64.0	41.0	51.2	55.0	41.0	47.3	62.0	19.0	41.3
Average			59.5			73.3			69.7			65.3			61.2

NIGHT TURN.

Apr.	61.5	55.0	57.0	77.5	69.0	71.8	73.0	65.5	67.6	67.0	61.0	63.5	59.5	44.0	52.7
May.	68.0	60.0	64.1	82.0	70.5	77.1	77.5	68.0	73.4	74.5	65.0	70.3	71.0	60.5	65.3
July.	72.0	69.0	70.3	88.5	83.5	84.4	84.0	80.5	81.4	78.0	75.5	76.6	73.0	64.5	68.2
Aug.	76.0	67.0	72.3	93.5	80.0	86.9	85.5	78.0	83.1	82.5	73.0	77.9	76.0	57.0	68.3
Sept.	70.0	59.5	66.0	78.5	66.5	73.8	77.5	65.0	71.4	75.0	64.0	69.8	67.0	48.0	58.2
Oct.	69.5	52.0	60.4	76.0	60.0	69.2	74.5	59.5	67.2	69.0	57.5	65.6	72.0	44.0	55.2
Nov.	56.0	39.5	43.0	64.0	54.0	58.7	60.0	50.0	55.5	52.5	45.5	50.0	54.0	30.0	40.7
Dec.	45.0	36.0	40.2	62.0	47.0	53.5	59.0	42.0	50.0	54.0	42.0	47.7	62.0	18.0	38.7
Average			59.2			71.9			68.7			65.2			55.9

TABLE VI.—*Recapitulation of Operators' Daily Report, Year 1912*

All Scrubbers, Primary

DAY TURN.

Month.	Temperature Cooling Water.			Temperature Gas after Scrubbers.			Temperature Gas after Fans.			Temperature Gas after Theisens.			Temperature Atmosphere.		
	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.
Jan.	40.0	33.5	34.8	70.0	46.5	55.7	63.0	45.0	52.8	50.0	38.5	42.6	48.0	55.0	25.0
Feb.	38.5	33.5	34.6	63.5	45.5	53.3	56.5	43.0	50.0	48.0	45.0	43.0	51.5	45.0	29.0
Mar.	49.0	35.0	41.2	94.5	54.5	65.7	86.5	53.0	62.9	63.0	44.0	51.0	56.5	20.0	40.0
Apr.	61.0	48.5	54.9	85.0	63.0	72.4	75.0	59.0	67.6	68.0	59.0	62.0	76.0	38.0	58.2
May	71.0	58.0	64.0	93.0	73.0	81.2	93.0	72.0	77.0	85.0	65.5	71.1	83.0	83.5	68.9
June	75.5	69.0	73.9	93.0	83.5	92.6	90.0	81.0	87.8	82.0	75.0	81.1	80.5	82.5	75.3
July	81.0	70.0	75.1	100.0	83.0	91.3	96.0	81.0	87.5	87.0	75.0	81.4	87.0	70.0	77.6
Aug.	76.0	67.5	72.6	93.0	83.5	87.0	89.0	78.5	83.4	81.5	73.0	78.1	81.5	64.5	74.0
Sept.	75.0	60.0	70.7	92.0	67.5	83.5	87.5	65.0	80.2	80.0	65.0	76.2	84.0	52.0	72.7
Oct.	67.0	52.0	60.2	78.0	62.5	73.6	76.0	60.0	67.9	71.5	57.5	65.7	75.5	47.0	60.2
Nov.	55.0	40.0	47.5	79.0	53.0	65.4	71.5	48.0	61.4	62.0	46.0	54.7	63.0	35.0	47.6
Dec.	47.0	36.0	39.1	67.0	43.0	51.9	64.0	41.0	49.6	55.0	41.0	46.1	62.0	19.0	39.8
Average			55.8			72.8			69.0			62.8			55.7

NIGHT TURN.

Jan.	40.0	33.0	34.6	67.5	45.5	54.9	63.0	44.5	52.2	48.5	41.0	43.5	44.0	3.0	23.6
Feb.	39.0	33.0	34.6	62.5	45.5	52.8	61.0	42.5	48.9	49.0	40.5	43.1	44.5	7.0	26.8
Mar.	49.5	35.0	41.2	75.0	52.5	61.7	71.0	51.0	60.6	57.5	43.5	53.6	56.5	19.0	37.5
Apr.	61.5	49.0	55.0	84.0	60.0	71.6	76.0	57.0	66.9	67.0	55.0	61.8	65.0	33.0	52.5
May	72.0	59.0	64.0	86.0	70.5	79.6	82.0	70.0	74.9	78.0	65.0	70.4	77.0	52.0	62.5
June	77.0	68.5	72.2	97.0	84.0	89.8	90.0	80.0	85.0	83.0	75.0	78.3	74.5	52.0	62.6
July	80.5	70.0	74.9	98.5	83.5	91.2	99.0	80.5	87.5	87.5	75.5	81.4	79.5	64.5	71.7
Aug.	76.0	67.0	72.3	93.5	80.0	86.3	88.0	78.0	83.1	82.5	73.0	77.9	76.0	57.0	68.5
Sept.	74.0	59.5	68.1	90.0	66.5	82.4	85.5	65.0	79.2	80.0	64.0	75.3	75.5	48.0	66.5
Oct.	69.5	51.0	60.0	76.0	61.0	69.3	74.5	59.5	67.3	69.0	57.0	65.3	72.0	44.0	55.0
Nov.	55.0	39.5	47.1	72.5	54.0	64.4	71.0	50.0	60.9	62.0	45.0	54.1	63.5	30.0	44.7
Dec.	45.0	35.0	48.3	65.0	45.0	52.8	89.0	42.0	49.5	54.0	42.0	46.0	62.0	18.0	37.9
Average			55.2			71.4			68.0			62.6			50.8

TABLE VII.—*Difference in Temperature Between Cooling Water and Off-Coming Gas Scrubbers, Year 1912*

Month	Day Turn		Night Turn	
	All Scrubbers.	Cut-off Type.	All Scrubbers.	Cut-off Type.
Jan.	20.9	——	20.3	——
Feb.	18.7	——	18.2	——
Mar.	24.5	——	20.5	——
Apr.	17.5	16.3	16.6	14.8
May	16.8	15.2	15.2	13.0
June	18.7	——	17.6	——
July	16.2	15.6	16.3	14.4
Aug.	13.4	14.1	14.5	14.6
Sept.	16.8	8.1	14.3	6.8
Oct.	13.4	13.0	9.3	8.8
Nov.	17.7	16.6	17.3	15.7
Dec.	12.8	12.6	14.5	13.3
Average	17.3	13.8	17.0	12.6

During part of the year 1912 we used some of our original scrubbers and part of our remodeled water cut-off type. The difference in temperature is notable. The cut-off type is included in the first column, "All Scrubbers," and tends to lower the average temperature, while the second column shows the temperature using the cut-off type alone.

Daily Observations

A daily log sheet of all important temperatures, pressures, and general performance of the plant is maintained. The daily observations may be summarized as follows:

Temperatures of the gas in mains throughout the plant.
Temperature of the atmosphere. (Recording.)
Temperature of cooling water.
Temperature of the waste water from scrubbers, fans, and Theisens.
Weir readings on settling-basin and fan-house discharges.
Pressures of gas in mains. (Recording.)
Psychrometric readings. (Atmosphere, sling-sychrometer.)

In addition to the above, notation is made on the log sheet as to general operating conditions, such as the number of scrubbers, fans, and Theisens in operation; number of stoves and boilers on clean gas; and number of gas power and blowing engines operating. At the end of each month, the data are compiled and averages struck. Reports are submitted to the various department heads interested.

Sampling Gas for Analysis

An average sample of the gas delivered to the gas engines is drawn over a period of 7 hr. each day. This is transmitted to the laboratory for analysis, which is reported back along with the calculated heat value in

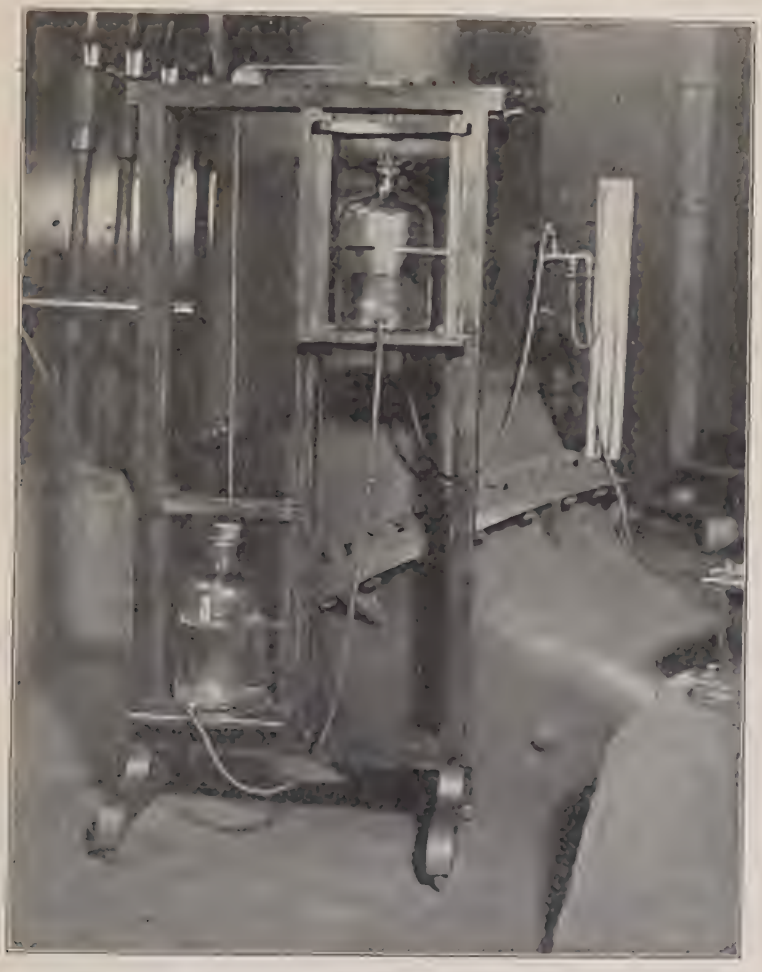

FIG. 13.—APPARATUS FOR TAKING GAS SAMPLES.

terms of net B.t.u. per cubic foot at 62° F. and 30 in. Hg. The apparatus used in drawing such sample, which we have found very convenient and entirely satisfactory for the purpose, is shown in Fig. 13.

Conducting and Reporting Tests

In conducting and reporting the various tests, we have endeavored to follow a standard method of procedure and form. A complete scrubber

D8033M616

CARNEGIE STEEL COMPANY—DUQUESNE WORKS

REPORT OF TESTS AT GAS CLEANING PLANT

Test No. 128 10/4 ,1910

DESCRIPTION OF TEST Dust determination on clean gas delivered to the gas engines.

MOISTURE AND DUST DETERMINATION—DATA

Gas Sampled at	#1 Theisen Separator.
Cubic Feet Gas Metered	313.991
Temperature " " °F	78.0
" " in Main "	76.6
" Inlet Water "	71.0
" Outlet " "	- -
Pressure Gas in Meter (Ins. Water)	-.55 Hg.
" " " Main " "	10.0
Barometric Reading (Ins. Mercury)	29.40
Temperature Atmosphere °F	- -
Weight of Dust as Sampled (Grammes)	.1791
" " Moist." " "	- -
Duration of Test	9:05 A.M. to 2:45 P.M.

2 Gas Power and 3 Gas Blowing engines operating.

RESULTS

Dust per Cubic Foot Dry Gas at	78.0	°F and	29.40	Ins. Mercury .00909	Grains
" " " " " " " "	76.6	" "	"	" " .00911	"
" " " " " " " "	62.0	" "	"	" " .00936	"
Moist." " " " " "	- -	" "	- - -	" " - - - -	"
" " " " " " " "	- -	" "	- - -	" " - - - -	"
" " " " " " " "	- -	" "	- - -	" " - - - -	"

RESULTS { 62 °F and 30.00" Mercury. / Grains per Cubic Foot Dry Gas } Moisture - - - - - Dust .00955:

VOLUME OF GAS—DATA

Reading taken at - - - - - - - -	Velocity Pressure, Pitot Tube, (Ins. Water) - - - - - -
Temperature of Gas in Main °F - - - - - -	Barometric Reading (Ins. Mercury) - - - - - - -
Pressure of Gas in Main (Ins. Water) - - - - -	Dry Gas Factor at 78.0 °F .968

RESULTS

Wet Gas per Min. at	- - - -	°F and	- - - -	Ins. Mercury - - - - -	Cubic Feet
Dry " " " "	- - - -	" "	- - - -	" " - - - - -	" "
" " " " "	- - - -	" "	- - - -	" " - - - - -	" "
" " " " "	- - - -	" "	- - - -	" " - - - - -	" "

ANALYSIS OF GAS—

Gas Sampled at	Outlet #1 Theisen.
Carbon Dioxide, CO_2 (% by Volume)	11.8
Ethylene, C_2H_4 " "	0.0
Oxygen, O_2 " "	0.0
Carbon Monoxide, CO " "	20.4
Methane, CH_4 " "	1.4
Hydrogen, H_2 " "	4.4
Nitrogen, N_2 " "	62.0
Heat Value, Calculated B.T.U.	95.3

CALORIMETER DATA—

Temperature Room °F	- - - - -
" Gas "	- - - - -
" Exhaust"	- - - - -
" Inlet Water °F	- - - - -
" Outlet " "	- - - - -
Pressure Gas (Ins. Water)	- - - - -
Barometric Reading (Ins. Mercury)	- - -
Lbs. Water Heated	- - - - -
Cubic Feet Gas Metered	- - - - -
Heat Value, B. T. U. 62 °F and 30 Ins. Mercury	- - - - -

test consisting of 71 items is shown in Table VIII. This manner of reporting such tests and the data shown, seem to cover pretty thoroughly the performance of the unit, including those items of observation which are of paramount importance, and we have found it most valuable.

I believe some such method should be pursued in making all washer tests at the various plants and would suggest a form similar to the one here shown. It would be of considerable advantage to have all tests reported in such a manner that they become comparative, which, of course, was our primal object in this case.

There is shown, also, a standard form for reporting dust and moisture determinations on engine gas. We find that this supplies just the data, as to performance, that we desire.

TABLE VIII.—*Standard Form for Gas-Washer Test. Carnegie Steel Co., Duquesne Furnaces. Gas-Cleaning Plant.*

Item	DESCRIPTION		Tower Washer Number 4
	General.		
1	Date of test		7/19/1913
2	Duration of test	Hrs. Min.	:46
3	Type of washer		Tower.
4	Inside diameter	Ft.	12
5	Height	Ft.	76
6	Area	Sq. Ft.	113.1
	Temperatures.		
7	Temperature of gas entering washer	F.	475.0
8	Temperature of gas leaving washer	F.	81.4
9	Temperature of water entering washer	F.	74.0
10	Temperature of water leaving washer	F.	112.4
	Pressures.		
11	Pressure in system before washer	Ins. Water	9.0
12	Pressure in system after washer	Ins. Water	5.0
	Quantities.		
13	Water consumed per minute in washer	Gallons	875.0
14	Volume of gas passing through washer per minute at 62° F. and 30 in. Hg.	Cu. Ft.	22,800.0
15	Weight of dust per cubic foot of dry gas at 62° F. and 30 in. Hg., entering washer.	Grains	2.6517
16	Weight of dust per cubic foot of dry gas at 62° F. and 30 in. Hg., leaving washer.	Grains	0.2257
17	Moisture per cubic foot dry gas at 62° F. and 30 in. Hg., entering washer.	Grains	38.4042
18	Moisture per cubic foot dry gas at 62° F. and 30 in. Hg., leaving washer.	Grains	11.7004
19	Moisture per cubic foot of gas at observed conditions, leaving washer.	Grains	10.2862
20	Moisture per cubic foot of gas, saturated at observed conditions, leaving washer.	Grains	11.4154
	Economic Performance.		
21	Volume of gas passing through washer per sq. ft. of area, at 62° F. and 30 in. Hg.	Cu. Ft.	20.16
22	Volume of gas passing through washer per gallon of water consumed, at 62° F. and 30 in. Hg.	Cu. Ft.	26.06
23	Difference in temperature between gas entering and leaving washer.	F.	393.6
24	Difference in temperature between water entering and leaving washer.	F.	38.4
25	Possible depression in gas temperature	F.	401.0
26	Actual depression in gas temperature	F.	393.6
	Heat Interchange.		
27	Heat in dry gas entering washer per minute (Above 32° F.)	B.T.U.	194,418
28	Heat in water vapor entering washer per minute (Total above 32° F.).	B.T.U.	160,090
29	Heat in dust entering washer per minute (Above 32° F.)	B.T.U.	579
30	Total heat above 32° F. entering washer per minute	B.T.U.	355,087
31	Heat in dry gas leaving washer per minute (Above 32° F.)	B.T.U.	20,690
32	Heat in water vapor leaving washer per minute (Total above 32° F.).	B.T.U.	41,697
33	Heat in dust leaving washer per minute (Above 32° F.)	B.T.U.	Nil.
34	Total heat above 32° F. leaving washer per minute	B.T.U.	62,387
35	Heat transferred to water and radiated	B.T.U.	292,700

TABLE VIII.—*Standard Form for Gas-Washer Test. Carnegie Steel Co., Duquesne Furnaces. Gas-Cleaning Plant. (Continued.)*

Item	DESCRIPTION		Tower Washer Number 4
36	Heat transferred to water as shown by observed rise in temperature.	B.T.U.	280,560
	Efficiencies.		
37	Efficiency of washer as dust remover	%	91.49
38	Efficiency of washer in lowering temperature of gas	%	98.15
	Gas Analysis, Gravity, etc.		
39	Carbon Dioxide, CO_2	% Vol.	12.8
40	Ethylene, C_2H_4	% Vol.	0.0
41	Oxygen O_2	% Vol.	0.0
42	Carbon Monoxide, CO	% Vol.	25.8
43	Methane, CH_4	% Vol.	0.0
44	Hydrogen, H_2	% Vol.	4.0
45	Nitrogen, N_2	% Vol.	57.4
46	Weight per cubic foot dry gas at 62° F. and 30 in. Hg.	Pounds	0.07651
47	Specific gravity referred to air	.....	1.0033
48	Calorific value of gas per cubic foot, gross, at 62° F. and 30 in. Hg.	B.T.U.	96.7
49	Calorific value of gas per cubic foot, net, at 62° F. and 30 in. Hg.	B.T.U.	94.7
	Moisture and Dust Determination.		
50	Gas sampled at	.....	After Washer
51	Volume of gas metered	Cu. Ft.	12.923
52	Temperature of gas at meter	F.	82.7
53	Pressure of gas in meter	Ins. Water	.....
54	Barometer reading	Ins. Hg.	29.18
55	Weight of dust as sampled	Grams	0.1708
56	Weight of moisture as sampled	Grams	8.2044
57	Dry gas factor at meter temperature	%	0.963
	Volume of Gas.		
58	Reading taken at	.....	.....
59	Temperature of gas in main	.....	81.4
60	Velocity pressure, Pitot Tube	Ins. Water	.....
61	Pressure gas in main	Ins. Water	5.0
62	Barometer reading	Ins. Hg.	29.18
63	Dry gas factor at main conditions of temperature	F.	0.963
64	Area of main at point where Pitometer is used	Sq. Ft.	.....
65	Velocity of gas per minute at point of measurement	Ft.	.....
66	Volume of gas per minute at main conditions	Cu. Ft.	.....
	Miscellany.		
67	Pumps, power requirements, average for test	K.W.H.	.....
68	Washer, power requirements, average for test	K.W.H.	.....
69	Cost of power per kw-hr	Cents	.....
70	Cost of water per 1,000 gallons	Cents	.....
71	Water pressure required for washers	Lb. per sq. in.	45 to 50

Moisture and Dust Determinations

Until within the last six or seven months, all moisture and dust determinations were made with the Brady apparatus. We found that the results on dirty gas could not be depended upon and were very inconsistent at times, and therefore adopted the method of dust estimation in dirty gas mentioned elsewhere in this paper. On clean gas, however, the results obtained by the use of the Brady apparatus are sufficiently reliable for all practical purposes. We are using the apparatus as originally designed, but we understand that it has since been considerably improved and that more reliable and accurate determinations are possible. Some few months ago a number of determinations of dust and moisture, Table IX, were made with the Brown apparatus, the results of which indicated accuracy and reliability. On engine gas we seem to get very good results with the Brady apparatus, encountering very little difficulty

and inconvenience in manipulation, at the same time taking every precaution to minimize error.

In making dust and moisture determinations it has been our aim to sample the gas at the same velocity as the gas flowing in the mains. With our present Brady apparatus, we find this a little tedious on clean gas to stoves and boilers, requiring constant attention and adjustment on the part of the observer. However, on engine gas we encounter very little inconvenience from this source and have been able to maintain the proper velocity with comparative ease.

The first dust and moisture determinations were made using volumetric tanks in determining gas volumes, which we found to be very inconvenient and decidedly limited. These we have replaced by wet test meters and as a result have sampled as much as 500 cu. ft. during a single test.

We have not made regular or daily determinations of dust and moisture, since we have not been inconvenienced by excessive dust deposition, either in our clean mains or the engine cylinders. Tests indicate a very uniform dust content in clean gas, with the moisture

TABLE IX.—*Résumé of Dust Tests Run with Brown Apparatus*

Date of Test	Cu. ft. Gas Metered	Tempt. Gas at Meter	Tempt. Gas at Main	Barometer	Press. Gas in Main	Duration of Test	Dust as Weighed	Dust per cu. ft. Dry Gas at 62° F. and 30 in. Hg.	Point at Which Sample was Taken
		° F.	° F.	In. Hg.	In. Water	Minutes	Grams		
7/16/13	9.962	76.50	82.20	29.40	8.00	30.0	0.0995	0.166	After Scrubber
7/17/13	22.843	83.80	80.30	29.17	9.00	105.0	0.3440	0.259	
7/18/13	8.590	84.70	86.10	29.18	8.00	30.5	0.0910	0.183	
7/18/13	9.537	87.00	87.00	29.18	7.00	35.5	0.1122	0.204	
7/19/13	9.135	80.60	80.40	29.18	6.00	33.0	0.0845	0.209	
7/19/13	12.923	82.70	81.40	29.18	6.00	43.0	0.1545	0.205	
7/21/13	3.867	80.00	79.30	29.30	7.00	15.0	0.0725	0.317	
7/22/13	5.055	88.70	81.40	29.40	6.00	21.0	0.0780	0.289	
7/22/13	6.620	86.50	81.00	29.37	6.00	23.5	0.0903	0.248	
7/31/13	7.205	93.00	86.00	29.30	5.00	25.0	0.0678	0.160	
Average								0.224	
7/23/13	4.524	75.40	85.50	29.28	6.00	21.00	0.0635	0.235	Before Fan
7/24/13	6.776	78.00	97.40	29.20	7.00	26.0	0.0980	0.244	
7/24/13	5.098	77.00	102.00	29.20	6.00	18.5	0.0555	0.183	
7/26/13	2.180	77.00	83.60	29.37	5.50	11.25	0.0305	0.234	
Average								0.224	
7/28/13	6.114	90.00	86.25	29.39	10.00	18.0	0.0240	0.066	After Fan Separator
7/29/13	9.049	87.10	89.70	29.39	12.00	29.0	0.0305	0.058	
7/29/13	4.370	93.00	89.00	29.38	12.00	11.5	0.0215	0.087	
7/30/13	10.986	88.40	89.10	29.34	10.00	39.0	0.0425	0.067	
Average								0.0695	

practically corresponding to saturation at the off-coming gas temperatures, and at no time have we encountered mechanically entrained water in amounts to be detrimental.

Raw vs. Clean Gas

The Use of Raw Gas

The use of raw gas in stoves is conducive to low efficiencies. This is due primarily to the heavy coating of dust on the brickwork fusing in masses, thus preventing the absorption of heat, destroying the brickwork, and requiring constant attention to the removal of this material. During the periods of scraping out the combustion chamber the stoves are burning gas very inefficiently, and at such times as the stove is off for complete cleaning the furnace is deprived of one-fourth of its blast-heating capacity. The intermittent pressures also make it impossible to obtain a constant flue-gas analysis and allow either excess air or gas in the stack.

The use of raw gas in the boilers shows a similar low efficiency, but not so marked. It is customary to blow off the tubes every 24 hr., and while the blowing is going on much excess air enters the combustion chamber. From the time the tubes are cleaned they start collecting dirt again and the steaming power suffers accordingly. Intermittent pressure also affects the boiler with regard to flue-gas analyses similarly to the stoves.

The Use of Clean Gas

By the use of clean gas in the stoves we completely do away with cleaning stoves, have a constant pressure in the gas supply, and use an orifice plate over the air-inlet door, which gives us an extremely constant and economical flue-gas. The saving on clean gas over dirty gas on stoves with regard to iron was shown in detail by the author in a discussion of Mr. Forbes's paper on gas cleaning,[1] read before the Institute in October, 1913, and amounts to 15.91c. per ton of iron.

The use of clean gas in the boilers gives the same benefits in operation.

The Use of Refined Gas

The thoroughly cleaned gas is used only for engine purposes. A yearly average of efficiencies of our power engines, with the analysis of the gas, is given in Tables XI and XII. The engines are 42 by 60 twin tandem Snow engines, driving a 2,000-kw. generator at $83\frac{1}{3}$ rev. The quantity of gas is approximately 5,000 cu. ft. per minute, each engine of this size taking about the same amount as a stove.

[1] *Trans.*, xlvii, 396 (1913).

TABLE X.—*Extracts from Boiler Tests*

	Observations		Fuel—Unwashed Blast-Furnace Gas	Fuel—Washed Blast-Furnace Gas
1	Number of tests	.....	16	15
2	Average duration of tests	Hrs., Min.	6 : 53	7 : 12
3	Water-heating surface	# Sq. Ft.	2,790	2,790
4	Steam pressure, gauge	Sq. In.	129.1	121.8
5	Draft	Ins. W.	0.6956	0.8226
6	Temperature of stack gases	°F.	649.3	652.8
7	Temperature of fuel gas	°F.	376.3	101.7
8	Calorific value of gas per cu. ft. at 32° F. and 30 in. Hg	B.t.u.	113.07	106.86
9	Calorific value of gas per cu. ft. at 32° F. and 30 in. Hg.	B.t.u.	104.34	96.44
10	Quality of steam, corrected	Per cent.	97.01	97.32
11	Equivalent evaporation per 1,000 cu. ft. gas	Lb.	59.14	61.93
12	Equivalent evaporation per sq. ft. heating surface	Lb.	3.880	3.254
13	Equivalent evaporation per 1,000 B.t.u.	Lb.	0.5297	0.6190
14	Fuel gas at 32° F. and 30 in. Hg. per developed H.P.	Cu. Ft.	613.7	572.3
15	Builders' rating	B.H.P.	250	250
16	Horse-power developed	B.H.P.	313.83	264.16
17	Per cent. of rated capacity developed	Per cent.	125.53	105.60
18	Heat lost due to incomplete combustion	Per cent.	7.28	7.20
19	Heat lost due to waste gases	Per cent.	27.92	27.44
20	Heat lost due radiation and unaccounted for	Per cent.	14.32	6.31
21	Efficiency (heat absorbed by water)	Per cent.	50.48	59.05

Item No. 8 calculated from analysis; Item No. 9 as per calorimeter.

Notation.—The above tests were run on boilers of the same make, type, and rating. It will be observed that the boilers working on clean gas developed but very little more than the builder's rating, although working at a higher efficiency than those on unwashed gas. This is due to an insufficient gas supply, which is directly traceable to undersized mains and connections, the mains having been so designed as to take care of only 1,500 boiler horse power of boilers on clean gas. In this case, however, the operation of approximately 3,750 boiler horse power was attempted, resulting in a gas shortage for each of the respective units.

The above tests were also run under normal conditions and are therefore representative of working performance. It has been our experience recently that steam can be raised very quickly on washed gas, having gotten such boilers on the line at 140 to 150 lb. gauge from feed water at 70° F., in 30 to 45 min. Due to very uniform gas pressures, as mentioned elsewhere, the boilers on washed gas steam very regularly.

There are four blowers and two power engines, the gas cylinders being the same in both cases. Gas containing 0.009 grain of dust per cubic foot is thus shown to be sufficiently clean for operating purposes. The data given relate to electric power engines. No. 1 gas power engine started in January, 1910, operated to Dec. 1, 1913, a total of 28,988 hr. No. 2 gas power engine started in July, 1910, operated to Dec. 1, 1913, a total of 27,095 hr. Total hours operated equaled 56,083.

Average load per engine, 1557.5 kw.

Average cu. ft. dry gas at 62° F. and 30 in. Hg consumed per kw-hr., 175.32.

Average thermal efficiency at switchboard, 19.12 per cent.

Average thermal efficiency per indicated horse power, 27.14 per cent.

Average dust in gas = 0.00925 grain per cu. ft. of dry gas at 62° F. and 30 in. Hg.

Average dust in air = 0.0016 grain per cu. ft. of dry air at 62° F. and 30 in. Hg.

Engines operate on constant mixture at all loads. Mixture is approximately one volume of gas to one volume of air.

Dust carried into cylinders per cu. ft. gas consumed = 0.00925 + 0.0016 = 0.01041 grain.

Gas consumed per engine per hour = 175.32 × 1557.5 = 273,061 cu. ft. at 62° F. and 30 in. Hg.

Gas consumed by both engines since starting = 56,083 × 273,061 = 15,314,080,063 cu. ft. at 62° F. and 30 in. Hg.

Dust carried into cylinders since engines were started = 15,314,080,063 × 0.01041 = 159,419,573 grains = 22,775 lb. = 11.387 tons.

TABLE XI.—*Performance of Gas Power Engines, Year* 1912

	Legend		No. 1 Engine	No. 2 Engine
1	Total time operating	Hr.	8,313.90	8,542.38
2	Total power output	Kw-hr.	14,020,290	14,158,340
3	Average load	Kw.	1,704	1,627
4	Load factor	Per cent.	85.20	81.35
5	Total gas consumed	Cu. ft.	2,461,872,000	2,601,288,000
6	Gas consumed per kw-hr.	Cu. ft.	175.59	183.73
7	Gas consumed per i.h.p-hr.	Cu. ft.	120.51	126.10
8	Gas consumed per b.h.p-hr.	Cu. ft.	91.59	95.84
9	Calorific value of gas, average for year	B.t.u.	95.80	95.80
10	Heat consumed per kw-hr.	B.t.u.	16,822	17,601
11	Heat consumed per b.h.p-hr.	B.t.u.	11,545	12,080
12	Heat consumed per i.h.p-hr.	B.t.u.	8,774	9,181
13	Thermal efficiency at switchboard	Per cent.	20.28	19.38
14	Thermal efficiency at b.h.p.	Per cent.	22.04	21.07
15	Thermal efficiency at i.h.p.	Per cent.	29.01	27.72
16	Gas consumed per minute, average for year	Cu. ft.	4,935	5,075
17	Gas consumed per hour, average for year	Cu. ft.	296,100	304,500
18	Time operating	Per cent.	94.40	97.50
19	Time delay due to engine	Per cent.	2.79	1.52
20	Time delay due to engine, not required	Per cent.	0.91	0.64
21	Time delay due to other causes	Per cent.	1.80	0.34

Gas volumes and calorific values all at 62° F. and 30 in. Hg.

Within a few hours after starting, all exposed parts inside of the cylinders except the wearing surface become coated with a light brown deposit. It has been our experience that if oil can be kept away from these surfaces this deposit is only very slightly increased by further operation of the engines. In the combustion chambers, where there is never any oil present, the deposit is so even that we have operated cylinders for nearly four years without cleaning these surfaces, and consider that the presence of such an even coating is an advantage, having an insulating effect, and protecting the metal from the direct action of the hot gases. However,

TABLE XII.—*Résumé of Dust Tests on Blast-Furnace Gas Delivered to Gas Engines*

Test No.	Cu. ft. Gas Metered	Temp. of Cooling Water	Temp. of Gas at Meter	Temp. of Gas at Main	Press. of Gas at Meter	Press. of Gas in Main	Barometer	Weight of Dust as Sampled	Duration of Test		Dry Gas Factor at Meter Temperature	Grains Dust per cu. ft. Dry Gas at 62° F. and 30 in. Hg.
		° F.	° F.	° F.	In. Water	In. Water	In. Hg.	Gram	Hr.	Min.		
67	8.500	34.0	44.0	46.0	13.5	13.5	29.24	0.0025	.....		0.991	0.00435
68	110.000	38.0	48.5	44.5	2.0	14.0	29.20	0.0100	2	15	0.989	0.00139
69	139.173	36.0	50.3	45.6	2.0	14.0	29.48	0.0430	3	0	0.988	0.00477
73	145.073	36.5	39.5	44.6	2.0	14.5	29.23	0.0230	4	15	0.892	0.00238
76	142.312	34.0	45.0	55.2	2.0	15.0	29.38	0.0505	3	15	0.990	0.00543
78	86.108	34.0	41.8	42.1	.....	14.0	29.54	0.0255	2	0	0.992	0.00449
79	83.561	33.0	44.0	40.2	.....	15.0	29.43	0.0360	2	15	0.991	0.00659
80	246.436	34.0	46.0	41.4	.....	13.5	29.31	0.1380	5	45	0.990	0.00865
81	116.561	34.0	50.2	44.5	.....	12.0	29.27	0.0405	3	0	0.988	0.00544
83	111.413	33.0	49.9	44.0	.....	14.0	29.35	0.0290	2	20	0.988	0.00406
85	117.020	33.5	44.6	41.0	.....	15.5	29.02	0.0435	2	45	0.990	0.00578
88	146.597	35.5	56.6	46.0	.....	14.5	29.07	0.0316	3	0	0.984	0.00343
89	139.421	36.0	38.0	45.5	.....	15.0	29.21	0.0144	3	0	0.992	0.00157
91	173.977	44.0	57.1	53.7	.....	14.5	29.39	0.1185	4	0	0.984	0.01079
92	132.570	45.0	53.3	50.0	.....	11.5	28.93	0.0465	3	30	0.987	0.00558
93	105.821	45.0	57.2	51.6	.....	13.0	29.15	0.0318	3	0	0.984	0.00485
94	83.669	44.0	57.3	52.0	.....	13.5	29.29	0.0070	2	15	0.984	0.00133
95	146.083	40.0	53.2	49.2	.....	14.5	29.27	0.0795	3	45	0.987	0.00856
96	141.358	41.5	66.8	52.7	.....	14.0	29.38	0.0410	3	0	0.978	0.00471
98	174.349	46.0	58.4	55.0	.....	14.0	29.41	0.0630	2	45	0.984	0.00573
99	81.684	48.0	74.0	60.0	.....	14.0	29.36	0.0153	2	15	0.971	0.00311
101	160.867	55.0	72.6	66.0	.....	15.5	29.36	0.0675	3	30	0.972	0.00690
102	109.836	53.5	68.0	62.0	.....	13.0	29.30	0.0865	2	50	0.977	0.01286
103	134.334	57.0	78.6	66.0	.....	16.0	29.23	0.0510	3	0	0 966	0.00648
105	95.325	54.0	50.1	59.1	2.0	12.0	28.81	0.0712	3	45	0.988	0.01183
106	149.865	62.0	78.8	68.0	0.50	10.5	29.29	0.1233	3	5	0.967	0.01386
107	158.331	59.5	63.5	67.0	0.32	11.0	29.45	0.0975	4	0	0.980	0.00991
108	173.734	61.0	63.1	67.4	1.28	14.0	29.48	0.1545	3	30	0.980	0.01422
109	169.426	59.0	70.4	65.2	0.53	12.0	29.43	0.0810	3	15	0.975	0.00783
110	173.499	61.0	69.9	68.5	0.46	13.0	29.02	0.0372	3	30	0.975	0.00355
111	175.474	59.0	58.6	65.0	0.80	12.0	29.36	0.0890	3	15	0.983	0.00804
112	144.382	60.0	72.3	67.0	0.40	12.0	29.45	0.0745	3	30	0.973	0.00848
113	143.685	68.0	75.2	75.2	0.37	12.0	29.02	0.0786	3	0	0.970	0.00921
114	165.480	64.0	72.1	71.2	0.77	11.0	29.46	0.1500	3	0	0.973	0.01469
115	124.324	75.5	85.5	82.0	0.20	10.0	29.22	0.1075	3	10	0.960	0.01490
116	139.880	77.5	84.0	85.0	0.80	12.0	29.06	0.0560	2	18	0.961	0.00737
117	184.017	78.0	87.9	86.2	0.45	13.0	29.30	0.1156	3	45	0.956	0.01088
118	142.312	80.5	88.9	87.2	.....	8.0	29.25	0.1763	3	10	0.955	0.02196
119	116.136	80.5	87.0	87.5	.....	10.5	29.15	0.1653	2	40	0.957	0.02453
120	173.090	81.5	88.5	88.5	.....	10.0	29.16	0.1460	3	40	0.955	0.01472
126	126.462	.....	64.6	79.0	.....	10.0	29.37	0.2037	2	45	0.980	0.02602
127	293.884	74.0	71.5	78.6	.....	11.0	29.37	0.2876	4	40	0.974	0.01590
128	313.991	71.0	78.0	76.6	0.55	10.0	29.40	0.1791	5	40	0.968	0.00955
129	236.920	73.0	63.0	77.6	0.50	9.0	29.50	0.1201	4	45	0.980	0.00812
130	373.780	68.5	74.1	75.8	.....	15.0	29.10	0.3143	7	45	0.972	0.01407
131	220.231	41.0	45.5	47.5	.....	12.0	29.25	0.2305	4	0	0.990	0.01613
132	250.980	46.8	56.9	43.8	1.24	14.5	28.74	0.2441	4	0	0.985	0.01571
133	236.041	35.7	44.6	44.6	2.76	14.0	28.94	0.2034	4	0	0.990	0.01355
Average												0.00925

where an excess of oil is present, as on the inside of cylinder head directly below the piston rod (the oil being carried in by the rod), the dirt accumulates more rapidly and often very unevenly. If an engine is shut down and the head allowed to cool, then started and brought up to working temperature again, the contraction and expansion of the head breaks the accumulated oil and dirt loose in the form of scale or blisters, the edges of which are easily heated and cause premature explosions. Usually this scale will partly break off, due to these premature explosions, and trouble at this point usually ceases until the next time the engine is stopped and started. When this trouble occurs too frequently, the engine is shut down, the cylinder examined through the opening for air-starting check valve, and the deposit scraped off if possible. If too hard or considered too extensive an accumulation the head is pulled out and the surfaces thoroughly cleaned. Following this procedure, we have some cylinder heads on our blowing engines that have not been out for over 18 months. Previous to August, 1913, when new piston rods were installed in the two power engines, practically every cylinder on the two engines had been closed up for more than a year, although examined through the starting-valve opening at shorter intervals.

We have an advantage over engines with exhaust valves directly in the bottom of the cylinder in that the oil and dirt that accumulate here do not interfere in any way with the operation of the valve, but can be blown out through the drain valve, with which each end of each cylinder is equipped. These valves are held open for two full cycles every 3 hr.

Our experience indicates that with our type of engine we can operate from 12 to 18 months without removing heads for cleaning, the smaller particles of dirt being removed at lesser intervals, and also that the cleaning is necessary principally from the fact that oil is present. The portions free from oil are covered so evenly that the presence of the deposit is an advantage.

Comparison of Systems

For the purpose of comparing general systems of gas cleaning, it might be interesting to note, first, a proposed installation of Duquesne tower washers as suggested for two furnaces, washing only for stoves. This installation depends on the simple scrubbing of gas without fans and will only reduce the top pressure about 2 in. of water, delivering gas to stoves at 0.5 grain of water above saturation point and cleaned to 0.2 grain of dust per cu. ft. The water used is standpipe water at its cost, delivered to point of consumption and boosted 15 ft. by a centrifugal pump. Second, an installation of two 40-ft. scrubbers in series, taking advantage of the bosh water head if the plant construction justifies the same, or with just sufficient "boosting" to obtain a 30-ft. head. These

installations will compare in cost per 100,000 cu. ft. of gas washed and per ton of iron as follows:

	Cost per 100,000 cu. ft.	Cost per Ton Iron
First case	0.04941	0.02386
Second case	0.00709	0.0034

It is evident that where sufficient head is available to use bosh water direct, two of these towers in series will make the cheapest installation possible, both as to first cost and operation cost, as the cost of pumping water is the major expense. The cost of operation between the towers and mechanical washers depends on the relation of water cost to power cost and machinery repairs.

It is also very desirable to have the temperature of the gas as low as possible to remove moisture, and this means either a series of machines or a counter current effect such as is obtained in a tower construction. It is also evident that in a machine where the water is in continuous contact with the gas, the gas will leave at the temperature of the outgoing water. The plant shown in Fig. 14 will be sufficient for two furnaces and will allow ample spare capacity, as each scrubber will be capable of handling 40,000 cu. ft. of gas per minute when pushed. This means that one scrubber can be used as a spare if so desired. An estimate of the cost of construction for the plant shown in the first proposition is as follows:

Estimated Cost of Duquesne Gas Scrubber for a Two-Furnace Plant

Excavating and foundations	$1,200
Scrubbers, incl. water valves and flues, main gas flues not included	13,450
Brick lining for flues, main gas flues not included	500
Supports for water valves	750
Pipe line, 4 in. in diameter	400
Rotary water valves	2,750
Water strainers	500
Centrifugal pump and motor, capacity 5,000,000 gal. per 24 hr	1,550
	$21,100
General expense and contingencies, 5 per cent	1,055
Total	$22,155

Cost of Cleaning Gas

The cost of cleaning gas for the year 1912 is given in Tables XIII to XV, and may be useful in showing an actual condition and what can be expected from an equipment working under the local conditions as outlined.

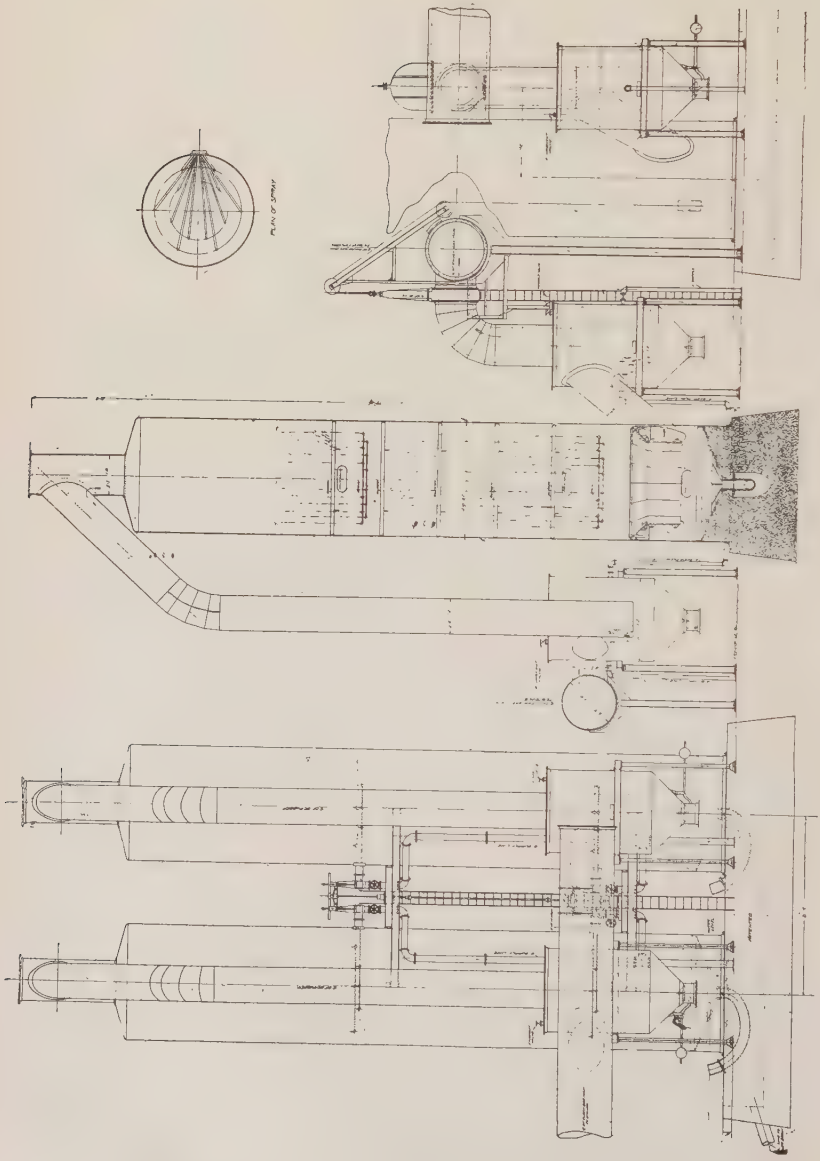

FIG. 14.—PROPOSED INSTALLATION OF SCRUBBERS FOR STOVES AND BOILERS FOR TWO FURNACES. CAPACITY OF EACH SCRUBBER, 40,000 CU. FT. PER MINUTE.

TABLE XIII.—*Cost of Cleaning Blast-Furnace Gas—Primary, Year 1912*
General Statement

		Year's Average
1	Gas cleaned:	
	No. 3, 4, 5, and 6 furnace stoves	3,117,660,000
	Gas engines	804,322,000
	Total	3,921,982,000
2	Water consumed (gallons):	
	Scrubbers	134,209,000
	Fans	22,157,000
	Water valves	13,869,000
	Total	170,235,000
3	Electric current consumed:	
	Lighting, kw-hr	2,203
	Pumps	51,650
	Fans	225,515
	Total	279,368
4	Producing labor cost	$373.42
5	Labor in repairs and maintenance	248.65
6	Materials in repairs and maintenance	190.65
7	Cost of electric current consumed	1,053.44
8	Cost of water consumed	595.88
9	Total cost	$2,462.04

Primary Cleaning—Economic Performance

		Year's Average
	Cu. ft. gas cleaned:	
1	Per gallon of water, scrubbers	29.65
2	Per gallon of water, fans	180.65
3	Per cu. ft. of water, scrubbers	222.26
4	Per cu. ft. of water, fans	1,354.11
5	Per gallon of water, primary	23.00
6	Per cu. ft. of water, primary	172.20
7	Per kw-hr., primary	14,615.70
	Cost per 100,000 cu. ft.	
8	Producing labor	$0.01052
9	Labor in repairs and maintenance	0.00630
10	Materials in repairs and maintenance	0.00481
11	Electric current	0.02660
12	Water	0.01506
13	Total cost per 100,000 cu. ft., primary	0.06329
	Percentage of total cost:	
14	Producing labor	16.62
15	Labor in repairs and maintenance	9.95
16	Materials in repairs and maintenance	7.60
17	Electric current	42.03
18	Water	23.80
19	Cost per 150,000 cu. ft. gas, primary	$0.09496
20	Cost per 1,000,000 B.t.u., primary	0.00657
21	Cost per 1,000,000 B.t.u., natural gas	0.04155
22	Cost per 1,000,000 B.t.u., coal	0.04464

TABLE XIV.—*Cost of Cleaning Blast-Furnace Gas—Final Cleaning, Year 1912. General Statement*

		Year's Average
1	Gas cleaned, engines	804,322,000
2	Water consumed, Theisens (gallons)	15,043,000
3	Electric current consumed:	
	Lighting, kw-hr	513
	Power	137,653
	Total	138,166
4	Producing labor cost	$157.50
5	Labor in repairs and maintenance	66.06
6	Materials in repairs and maintenance	140.80
7	Cost of electric current consumed	538.54
8	Cost of water consumed	52.65
9	Total cost	$955.55

Final Cleaning—Economic Performance

		Year's Average.
	Cu. ft. gas cleaned:	
1	Per gallon water consumed	53.48
2	Per cu. ft. water consumed	400.77
3	Per kw-hr. water consumed	5,822.10
	Cost per 100,000 cu. ft.:	
4	Producing labor	$0.01965
5	Labor in repairs and maintenance	0.00822
6	Materials in repairs and maintenance	0.01701
7	Electric current	0.06710
8	Water	0.00658
9	Total cost per 100,000 cu. ft., final cleaning	0.11856
	Percentage of total cost:	
10	Producing labor	16.57
11	Labor in repairs and maintenance	6.94
12	Materials in repairs and maintenance	14.35
13	Electric current	56.59
14	Water	5.55
	Cost per 1,000,000 B.t.u.:	
15	Final cleaning	$0.01237
16	Natural gas	0.04155
17	Coal	0.04464
18	Cost per 150,000 cu. ft.	0.17784

TABLE XV.—*Cost of Cleaning Blast-Furnace Gas*, 1912. *General Exhibit*

A.	Cost per ton of iron—Gas cleaned and delivered to stoves	Average for Year
1	Total gas delivered to stoves for month, cu. ft.	32,406.93
2	Cost per 100,000 cu. ft. at stoves, primary	$0.06329
3	Total cost gas cleaned and delivered to stoves	$2,051.03
4	Tonnage, furnace working on clean gas in stoves	63,061
5	Cost per ton of iron—gas cleaned and delivered to stoves	$0.03252

B.	Cost per ton of iron—Gas cleaned and delivered to blowing engines	Year's Aver.
1	Total gas delivered to engines for month, cu. ft.	3,828.01
2	Cost per 100,000 cu. ft., gas delivered to engines	$0.18185
3	Total cost of gas cleaned and delivered to engines	$696.124
4	Tonnage credited to gas-blowing engines	31,524
5	Cost per ton of iron—gas cleaned and delivered to engines	$0.022082

C.	Cost per 1,000 kw-hr., gas cleaned and delivered to gas power engines	Year's Aver.
1	Total gas delivered to gas power engines for month, cu. ft.	4,215.21
2	Total cost 100,000 cu. ft.—gas delivered to engines	$0.18185
3	Total cost gas cleaned and delivered to engines in month	$766.536
4	Total power output, gas power engines—kw-hr.	2,367,386
5	Cost per 1,000 kw-hr.—gas cleaned and delivered to engines	$0.32379

Note: Items No. 1—100,000's cu. ft.

Cost of Fans and Scrubbers

The fan operation is expensive, due to the high power consumption. This can be noted in the yearly cost for 1912. If we had ample mains and good draft, as can be obtained in a simple plant for only scrubbing for stoves and boilers, a fan would be unnecessary and costs would be lessened accordingly. If bosh water were used at no charge, the cost would be lowered further. The cost of water and power in 1912 was 66 per cent. of the entire cost, and with a simple tower plant, without fans and using bosh water, the cost would be reduced in that proportion. The operating force due to the secondary cleaning, handling of motors, etc., gives a proportion chargeable to primary cleaning which would be unnecessary in a simple primary tower plant for stoves and boilers, as the attention so required is very small. From the costs shown and the deductions which may be drawn, it is very reasonable to expect to wash gas by efficient towers sufficiently clean for stoves and boilers for 1.5c. per ton of iron.

Cost of Theisens

The Theisen cost is high, due also to power and service water. The power cost cannot be changed very much with this type of machine. The water expense would be cut down if bosh water were used. The repair cost is high, as is natural to expect in such a rapidly revolving piece of apparatus.

Conclusion

There are numerous ways and means of cleaning gas, as presented in Mr. Forbes's paper before the Institute in October, 1913. The dry cleaning, as at present operated, is cumbersome, occupies large space, is expensive to install and is open to numerous operating difficulties. In all wet systems the handling of the fine sludge becomes the same problem, whether it is removed by mechanical devices or by towers, and from the standpoint of an operator, the simplest device is generally the most inexpensive and requires the least attention. From the experience derived from our installation it seems that a simple tower with proper means of preventing channeling is the easiest solution for primary gas, as 0.20 to 0.25 is generally conceded to be sufficiently clean for stove and boiler purposes.

The author wishes to gratefully acknowledge the aid of S. E. Eldridge, in charge of gas cleaning at the Duquesne Works, under whose direction many of these data were collected and tests prepared. Other data, tests, and deductions, as well as some of the details only outlined here, were presented to the Institute by the author in a discussion of Mr. Forbes's paper on gas cleaning in October, 1913, and might be interesting in this connection. It has been the desire in the present paper to present actual and experimental data, as well as general and interesting information on the subject of blast-furnace gas cleaning. The data are confined to one plant, but as that plant has operated for five years, it is representative of general practice, and the information presented may be useful as an aid to others in furthering the knowledge on this present live subject.

Discussion

A. E. Maccoun, Braddock, Pa.—To the practical blast-furnace man this paper is of the greatest interest. Every furnace manager in this country is striving to procure a better product and decrease the cost of production. One important factor in obtaining these results is the maximum utilization of the energy contained in blast-furnace gases.

At the Edgar Thomson Works of the Carnegie Steel Co. we installed, in 1906, primary washers to wash the gas supplying the stoves of four blast furnaces, and in addition to this to furnish enough primary washed

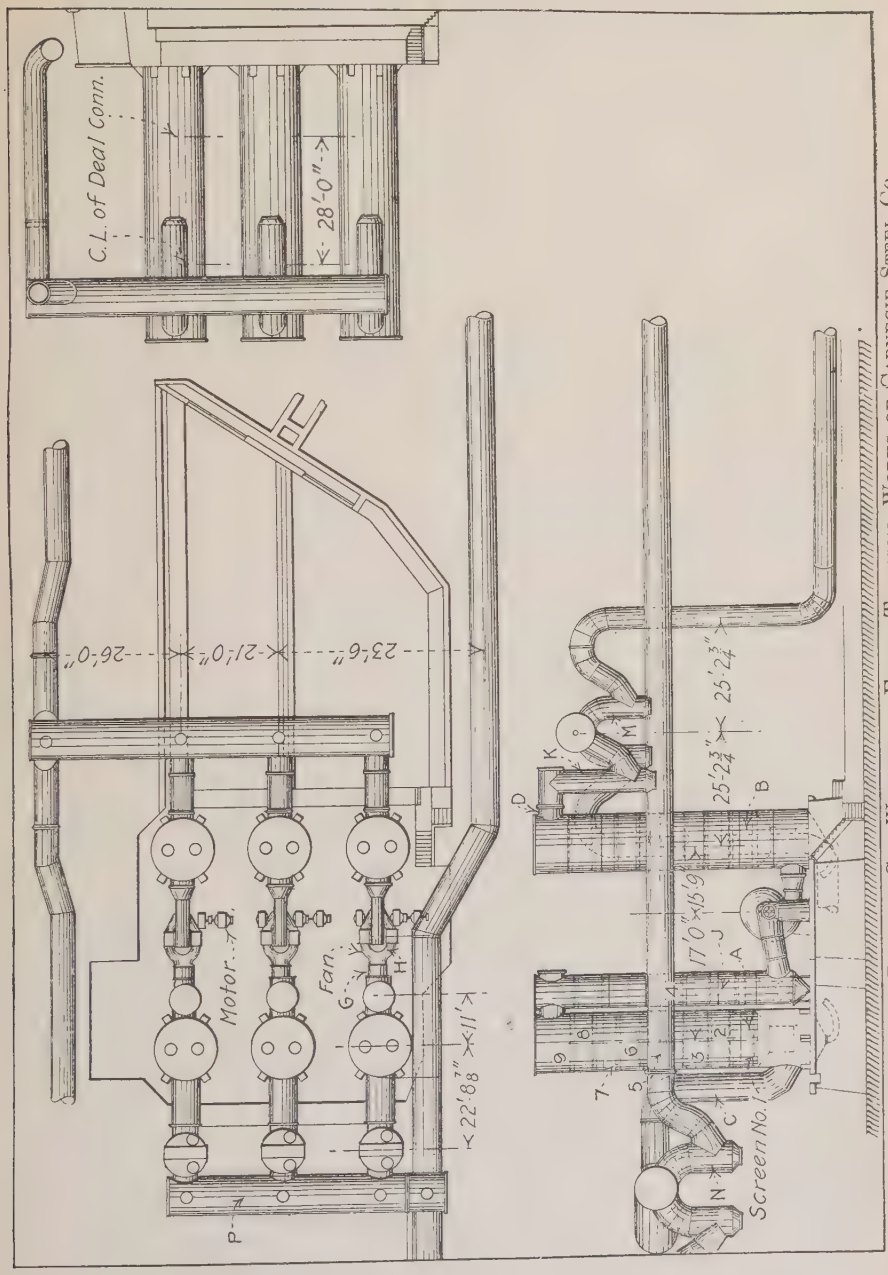

FIG. 15.—PRIMARY GAS WASHER AT EDGAR THOMSON WORKS OF CARNEGIE STEEL CO.

gas to supply Theisen washers for three Westinghouse 3,000-h.p. gas engines. We are now installing additional primary washers to wash the gas for the stoves of all the 11 blast furnaces located at this plant. Fig. 15 gives the general outline of one of these newer gas-washing installations.

Being intimately connected with this work since its inception, as well as being familiar with the system installed at the Duquesne plant, I wish to call attention to the valuable data Mr. Diehl has submitted to this Institute, before proceeding with the discussion. These data represent extensive tests covering five years of operation, and were obtained only after a large amount of costly research work, and are therefore very valuable to any one interested in the subject. In fact, I do not know that such complete data on this subject have ever before been published.

Mr. Diehl's tests show the absolute necessity of obtaining clean gas for use in blast-furnace stoves if we expect to secure the most efficient results.

Different Systems of Gas Cleaning

At the October, 1913, meeting of the Institute Mr. Forbes read a paper describing the different systems of gas cleaning in use. There are two distinct cleaning processes, wet washing and dry cleaning.

The usual practice for a plant with several blast furnaces is to use an efficient dust catcher on each furnace and at times additional auxiliary dust catchers, or dry cleaners, so as to catch as much as possible of the heavy material sent over with the gases, as this material can be better taken care of in the dry state. From here the gas is collected in one central dirty-gas main and conducted to the additional cleaners; or it can be used directly as dirty gas in emergencies.

This arrangement tends to equalize the variations in the amounts of gas given off by the different furnaces, and better and more regular results are obtained from the cleaning plants.

Many attempts have been made to discover a practical dry-cleaning or filtering process which would clean gas sufficiently to allow it to be burned in the blast-furnace stoves with the best possible results. The sensible heat of the gas which is lost by washing amounts to about 4 per cent. of the total heat value of the gas, assuming that the unwashed gas, by the time it reaches the stove or boiler burners, is at very nearly 300° F., which is approximately its true temperature.

There are numerous objections to most of the proposed dry-cleaning processes. Of course, with gas engines, it is absolutely necessary to cool the gases, so as to have the smallest possible engine cylinders and accurate ignition. Both of these results are best obtained with dry, cool gas.

Some have tried to combine the wet and dry systems by directing hot gases against water surfaces. Most of these systems, if they are very efficient in the removal of dirt, are objectionable, on account of the outgoing gas becoming laden with moisture.

We are at present experimenting on a cleaner of this type modeled after the smoke consumers at the New York Edison power plant; it is known as the Murray cleaner. The gas is taken through a restricted orifice and injected against a diagonal plate on which a film of water is always running; it is a very simple device and the water consumption is low. On test this device has shown up remarkably well in that the gas loses very little heat in passing through the cleaner, and the outgoing gas is remarkably clean and low in moisture. We have placed this cleaner on one of our hot-blast stoves, and will try it by actual use over a long period, and thereby demonstrate its real value as a device for this class of work.

The gas for this cleaner on the test referred to was taken off the main dirty-gas flue connecting all furnaces. Inlet gas on an average contained 15.342 grains of moisture per cubic foot and 0.9210 dust; outlet contained 4.414 grains of moisture and 0.2252 dust per cubic foot.

The Cottrell process of electrical precipitation has also been mentioned in this connection. I understand the Bethlehem Steel Co. are to try this out on a small scale. There will no doubt be other developments along this line. It is to be hoped that some safe, efficient dry-cleaning process will finally be developed.

We have had very little experience in this country with primary mechanical washers, except in utilizing this type of washer in the further refining of gas for gas-engine use, and therefore are not in a position to criticise these primary washers to any great extent. They have not been looked upon favorably, however, on account of their power requirements and the fear of large maintenance costs, etc.

Tower washers of the baffle, Zschocke, and spray types are used to a large extent for primary cleaning at our blast-furnace plants on account of their simplicity and the absence of moving parts; all these washers give excellent results. At Edgar Thomson our original washers were of the baffle type, but it was found that the baffle plates used in the towers rapidly became built up in spots with flue dust, thus causing the gas to "channel" through narrow passages, and therefore the efficiency of the baffle was not as high as it should be. Aside from this, the water did not form a very uniform film over the edges of the baffles, so that the contact of the gas and water was not intimate enough. We are therefore abandoning this type of tower washer in favor of the Diehl spray type.

Diehl Spray Type of Gas Washer

Additional advantages of this type of tower washer are:

The rotary valve is on the outside of the tower and is easily reached for repairs. Some of the other types of rotary spray washers require taking the gas off the towers to make these repairs.

The area through the washer is not as restricted as through the baffle type, as this type of washer has the advantage of the entire area of tower, whereas the inner tube in the baffle type of washer takes up about half the area, and there is no cleaning done in this tube.

The counter-current principle is always most effective for the transfer of heat. Most of the various types of tower gas washers work on this principle.

The gas is introduced into the tower from below and ascends slowly, the water descending from above, so that the coolest gas always comes in contact with the coldest water and the hottest gas with the hottest water. The Diehl type of spray carries this principle even further, as the water, by means of pressure, is sprayed up a certain distance and then descends. Mr. Diehl's paper and tests show how efficient this intermittent-spray type of washer is, in that the difference in temperature between the gas leaving the washer and the ingoing water is approximately only 5° or 6° F.

In some cases fans are used to boost the pressure; in other cases the pressure from the furnace is sufficient. At Edgar Thomson we use fans between the towers, but very little, if any, cleaning is attributed to the fans, as no water is run on them at the present time. When they were originally installed a series of Koerting sprays were used and water was run on them continuously. This caused the fan rotors to eat away and the power required to drive them was the heaviest item entering into the cost of cleaning. Spray pipes at high water pressure are connected to them twice a week for a short time only, and the sides of the casing and rotor are thoroughly flushed out.

Mr. Diehl gives an interesting example of a proposed installation of his type of tower washer for two furnaces, washing only for stoves, which depends on the simple scrubbing of the gas without fans. It can be readily seen how well this plan works out and how low the cost of construction ($22,155) for such a plant would be for the saving effected.

Moisture

I have known of some cases where gas-washing plants have been installed and very bad results were obtained by using the gas on the hot-blast stoves. In nearly all such cases it developed that the gas was not being cooled sufficiently to precipitate the moisture and eliminate it from the gas. The moisture must be eliminated to as great an extent as possible; the gas should be cooled below 70° F. to obtain good results.

Fig. 16 shows the moisture curves of blast-furnace gas, giving at the various temperatures the grains of water present per cubic foot of gas reduced to standard dry gas at t° and 30 in. mercury.

Curve No. 1.—So-called "saturated gas"; *i.e.*, gas mixed with saturated

aqueous vapor. At and above 212° F., its value is infinity at atmospheric pressure.

Curve No. 2.—Moisture accompanying each cubic foot of dry standard gas produced from normal unwatered furnace stock.

Curve No. 3.—Moisture accompanying each cubic foot of dry standard gas produced from normal furnace stock to which has been added 6 per cent. of its weight of water to thoroughly moisten it and reduce the amount of flue dust.

At temperatures below the intersections of curves Nos. 2 and 3 with the "saturation" curve, No. 1, a considerable amount of the accompanying moisture of these gases cannot be held as true vapor, and is either held to some extent in a supersaturated state, or is precipitated out.

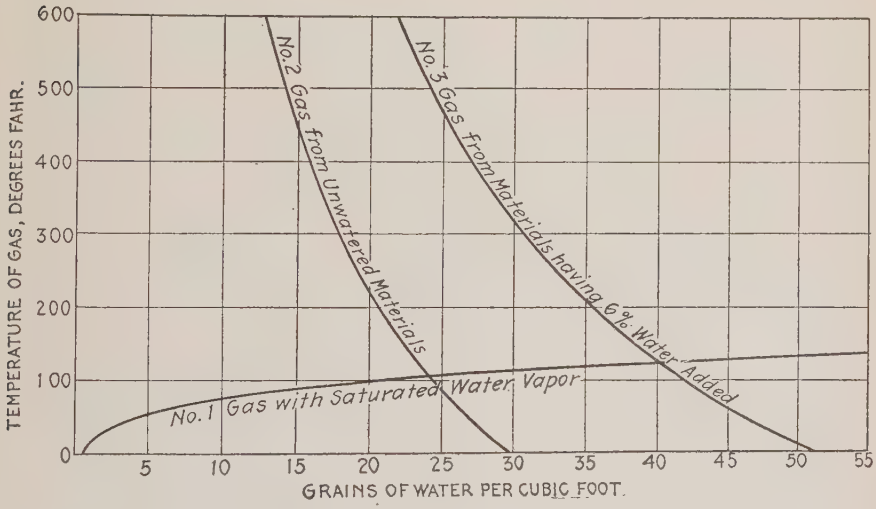

FIG. 16.—MOISTURE CONTENT OF BLAST-FURNACE GAS, CALCULATED TO DRY GAS AT t° AND 30 IN. MERCURY.

Study of these curves shows that the best results cannot be obtained from washed gas if it is not cooled sufficiently to obtain low moisture content.

One of the most important considerations in the use of clean gas is to be able to mix the proper amount of air and gas at the stove burner and thereby obtain perfect combustion. Mr. Diehl partly shows the efficiency of dry cool gas in his tables, but he fails to state that it is impossible to burn dirty gas efficiently in any case, because if it is attempted the high temperature of combustion melts some of the flue dust carried by the gas down into the well of the stove, filling it up, and gradually forming a big solid lump of sinter, which it is nearly impossible to remove without the expenditure of a large amount of labor and a great loss of time. To

avoid this when using dirty gas a low temperature of combustion is purposely sought in the combustion chamber of the stove at the expense of loss in efficiency.

Nitrogen Compounds

It is fair to ignore the formation of nitrogen compounds in the gas, as Mr. Diehl's experiments have failed to even find traces of these compounds upon special research analyses. The coloration observed at the washers could be readily produced by extremely minute traces of ferrocyanides.

The Efficiency of Hot-Blast Stoves

The importance of high heats is fully realized in modern blast-furnace practice and we wish to obtain these higher heats by utilizing blast-furnace gas as efficiently as possible, as the demand for the gas remaining after supplying the stoves and blowing engines is becoming greater all the time. Steel works are endeavoring to turn out better products, which necessarily require more refining and work, and an increasing amount of power. Blast-furnace gases have a much greater fuel value than an amount of coal containing an equal number of B.t.u.'s, as they require no expensive handling and make an ideal fuel for nearly any use around a steel plant.

The stove tests submitted by Mr. Diehl show conclusively the better efficiencies obtained by the use of washed gas, but in comparing these results it must be borne in mind that the stoves burning the clean gas on which tests were made were not originally built for this gas. They should have had much smaller checker openings; thereby much greater heating surface would have been obtained and their efficiency greatly increased over the figures presented. With dirty gas, where it is not possible to use small checkers successfully, the use of a 9-in. checker opening for stoves is common. On our newer stoves, adapted for clean gas, the checker openings have been decreased to 6 in., and in some cases to 5 in., the heating surface gained being approximately 30 per cent.

In this connection I wish to refer to another point that has not been brought out fully enough. With a *thoroughly cleaned stove* good results can be obtained for a short time with dirty gas.

Take a blast-furnace that has four clean stoves and uses dirty gas. At first excellent heats are obtained; gradually the checker openings begin to clog with fine dust; it bridges over the top of the checkers, the heats begin to run down quickly, and the efficiency drops. I have known of such cases where the decrease in stove efficiency has been over 12 per cent. The coke consumption of the furnace then increases, the product decreases and the furnace practice becomes very poor. Then one stove at a time is taken off for cleaning, the results being still poor until the stoves are

finally all cleaned, and we have the four stoves again in use. During this time the furnace has become accustomed to low heats, and it usually takes some time to bring it back to the former good practice. When this is finally accomplished it is not long before the operation has to be repeated and the stoves cleaned again. Therefore the stove tests do not represent at all the actual gain in blast-furnace practice obtained from the use of clean gas, as the greatest gain is having a regular, dependable hot-blast temperature and the elimination of as many variable furnace conditions as possible.

In addition to the above, the use of clean gas represents a large saving in repairs and renewals required on the stove linings, etc.; also on hot-blast valves and seats, pipes, tuyères, etc., as these are very badly cut and costly repairs are continually required when dirty gas is used.

We frequently find the bustle pipe half filled up on one side of a furnace that has been on for any length of time, which also tends to irregular distribution of blast.

All of these important factors must be considered in arriving at the true gain represented by the use of clean gas for stoves.

There are further economies to be made which will also increase the efficiencies obtained in hot-blast stoves, and which I wish to mention in this connection:

1. The tables Mr. Diehl has submitted and our own tests show the heat loss in flue gases from our hot-blast stoves to be approximately 20 per cent. This heat should be utilized to add to the temperature of the cold blast before it enters the stove to be further heated. I understand this is done at some of the German blast-furnace plants.

2. Radiation losses, figuring roughly on the exposed surface of the stove shell, vary from 7 to 12 per cent. This is hard to arrive at accurately, as no radiation formulas are available to suit this particular condition. By more efficient insulation between the brickwork and the shell a saving might be effected here. At present we have a 2-in. space filled with loam mixed with granulated blast-furnace cinder.

3. A more efficient burner will be devised to get as high a combustion temperature as possible in the lower part of the combustion chamber, and thus shorten the flame and confine all the combustion to the combustion chamber, so that the hot products of combustion alone will pass through the checker work. In most cases the combustion is not completed at the top of the checkers.

Savings Through Gas Cleaning on Stoves

At some blast-furnace plants hard ores or ores of good physical structure are used to a large extent in the burden. This, combined with slow driving, will greatly reduce the usual troubles in stoves caused by the flue

dust carried in the gas; but such cases are rare, and do not apply to modern blast-furnaces using a large percentage of Mesabi ore in their burdens, which furnaces we are here considering.

Mr. Diehl gives the cost of primary cleaning gas for stoves per 100,000 cu. ft. as $0.06329, or $0.03252 per ton of iron, which would be very much reduced with a new installation arranged so that not so much electrical power would be required for operating the fans.

The saving on clean gas over dirty gas on stoves with regard to the cost of iron is given as $0.16108 per ton of iron. This I consider very conservative; in fact, I consider that $0.22 per ton of iron is a more representative figure for this saving for a modern blast-furnace using a large percentage of Mesabi ore.

Savings Through Gas Cleaning for Boilers

No attempt has been made at the Edgar Thomson Works to clean gas for boilers except on tests, as the boilers at our plant give very little trouble from the dust in the gas building up in passages and tubes, and what does gather there is easily blown away by steam from a hose fitted with a 1-in. nozzle. We have made boiler tests using clean gas, but could not see enough saving in its use for this purpose to warrant the investment. The saving depends on what type of boiler is used.

F. Louis Grammer, Leesburg, Va.—I ask Mr. Diehl whether he has ever analyzed blast-furnace gas when the furnace was hanging—I do not mean hanging around the bosh, but hanging due to throttle near the top—or whether he ever analyzed the gas immediately preceding such a slip. I remember that in blowing in a furnace at Cleveland, owing to congested traffic conditions, the locomotive was not able to get around to dump the dust catcher. The result was that our blast pressure rose, consequent to back gas pressure, to some abnormal figure, such as 26 lb., and yet everything seemed to be all right. There was a good deal of concern over the situation, and no one understood what the cause of the trouble was until it was discovered that the dust catcher had not been dumped. When the dust catcher was dumped the pressure dropped to about 14 lb.

I have an idea, which has not been fully tested—I only offer it as a suggestion—that possibly a throttling of the escaping gases would tend to make for excessive carbon deposition in the upper part of the stack, and as a preliminary to such a congestion this excess would be reflected in the changed composition of the gas. This subject received a good deal of attention at my hands in 1904, when we had available nothing like such an elaborate presentation of the subject as Mr. Diehl has just given. I make that suggestion as of possible interest in case further investigations along this line are made. Personally, I believe that the throttling of the

downcomer would tend to decompose the escaping gases so that the carbon decomposition would be increased. My impression is that "spilling" occurs more frequently on dirty flues.

A. N. Diehl.—I do not recall any analyses that we have purposely taken before and after slips, with the object of determining the point which Mr. Grammer has in mind. As mentioned in my original paper, we did at one time take some analyses of gas directly after a slip to account for the back-firing of our gas engines. We suspected hydrogen in this gas due to moisture in the stock, and therefore prepared ourselves for this test. We have often noticed when a furnace becomes very tight and is hanging continuously, that the gas becomes very sharp, which can easily be noticed as it is burned in the stoves and boilers. This might account for the condition such as Mr. Grammer is referring to. I will be glad to make some experiments along these lines to obtain further information in the matter.

J. W. Richards, So. Bethlehem, Pa.—I ask Mr. Diehl whether, in the dry dust or washing water which may have been used, there is any concentration of potassium salts; that is, whether any of these have been detected in the dust, or in the water used for cooling.

A. N. Diehl.—I have shown in my paper analyses of the deposits sublimated on the ports and piston heads of gas engines, indicating a content of alkalies equal to 30 to 40 per cent. The water from the scrubbers is alkaline to a certain extent and we find soda and potassa present. We often find quite heavy deposits of cyanides and alkalies in the brick lining of the blast furnaces, some analyses of which show about 40 per cent. These deposits seem to permeate the brick.

J. W. Richards.—I asked the question because Mr. Eckles of Washington had patented the use of potassium-iron silicates in the blast furnace, in order to collect the potassium which would be volatilized and caught in the dust coming from the furnace. I wanted to know to what extent that worked out in ordinary practice.

A. N. Diehl.—On the fans of our Theisens we obtain deposits also of alkalies, but the quantity is comparatively small considering the large quantity of gas passing through. I do not have any figures to show the quantity obtained in this manner.

Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes

BY HEINRICH J. FREYN, CHICAGO, ILL.

(New York Meeting February, 1914)

THE American iron and steel manufacturer finds himself to-day barely at the threshold of enormous possibilities for practicing rational economy in the use of fuels. The fuel cost is by no means the smallest of items which enter into the cost of manufacture of iron and steel products, but the great wealth of this country in high-grade fuels, together with a protective tariff, has heretofore made fuel economy relatively unimportant.

In Germany, the use of raw coal for the purposes of iron and steel manufacture has been all but abandoned and large sums of money, formerly irrevocably wasted, are now being saved.

When scanning the literature of this country devoted to the application of gas power, but little information will be found on a subject which, however, is destined to become of considerable importance in the United States, namely, the use of coke-oven gas for the generation of power in gas engines.

This lack of information can be traced to the fact that the regenerative by-product coke oven with its large amount of surplus gas is still far from having attained in this country the universal application which it has found in Europe. This statement needs hardly any corroboration other than the evidence of statistical investigations made by E. W. Parker of the United States Geological Survey, according to which only approximately 25 per cent. of the total coke produced in this country in 1912 was made in by-product coke ovens, whereas 75 per cent. was produced in beehive ovens. This entailed the enormous loss of over $80,000,000 in non-recovered by-products, such as gas, tar, and ammonia, without considering the substantial loss due to the non-recovery of benzol from the coal.

There are two more or less distinct fields of application of the by-product coke oven: one, to furnish as the main product metallurgical coke of specified quality, and surplus coke-oven gas as the by-product, and the other, to carbonize high-volatile coals, and thus produce as the main product large quantities of gas, high in heat value and rich in illuminants, adapted primarily for domestic heating and lighting, while domestic coke becomes the by-product.

Owing to the large tonnage of coke required in metallurgical industries vast quantities of surplus gas become available, for which a field of usefulness must be found. Wherever a steel works is connected with the blast-furnace plant, this surplus gas can be used advantageously in addition to that derived from the surplus blast-furnace gas, in the manufacture and heating of steel, or for the production of power. Thus the necessity of shipping into the plant any other coal except coking coals can be practically eliminated.

With isolated blast-furnace plants which produce their own by-product coke, the uses of the surplus gas are not so obvious, since the quantity of surplus blast-furnace gas by far exceeds the immediate demand for power. But where blast furnaces are grouped in the vicinity of large cities, they will have a better opportunity than ever for profitably selling their surplus gases for domestic heating and lighting, or in the form of electric power, because they can furnish these commodities at lower cost than the cities themselves could produce them.

The modern by-product coke oven is fully capable of producing excellent metallurgical coke for blast-furnace use, from certain coals and coal mixtures, and, in addition, a quantity of surplus gas commensurate in every way with coal and water gas, and of a heating value equal to modern specifications for illuminating gas. By the proper selection of coking coals, the blast-furnace manager is thus placed in the enviable position of manufacturing blast-furnace coke, satisfactory to *himself*, in coke ovens operated under *his* own management, and yet of furnishing all, or at least the rich portion of, his surplus coke-oven gas for distribution through the city gas mains.

But a large field of opportunity remains to apply profitably the surplus gases of metallurgical industries for the purposes of neighboring communities, even in the steadily diminishing instances where the obsolete minimum-candle-power specification is still maintained, and where public service corporations prefer to install by-product gas chamber ovens of their own for the purpose of carbonizing gas coals and disposing of the total coke-oven gas for illuminating purposes. Modern industrial life causes the demand for electric light and power to increase by leaps and bounds, so that public service corporations are constantly obliged to add to the capacity of their electric light and power plants. The purchase by such corporations of electric current generated from industrial surplus gases at nearby blast furnaces and coke-oven plants, or the acquisition of these surplus gases for power production in power plants of their own, becomes a very desirable transaction, which offers financial advantage to both seller and buyer, and ultimately to the consuming public at large.

By taking advantage of the recent innovation of H. A. Brassert, blast furnaces are now able to insure fulfillment of a contract for the delivery of

a specified uniform amount of power in times of business depression as well as activity. Brassert virtually transforms a non-active blast furnace into a huge gas producer, and gasifies in it, small coke, raw coal, and coke breeze, on the principle of the slagging type of producer, using open-hearth slag as flux. This has been very successfully demonstrated at South Chicago and Gary.

This disposes of the argument that curtailment of pig iron and coke production in time of business depression militates against the fulfillment of a contract of this nature.

In order to show what can be accomplished in the practical utilization of the surplus gases of a blast furnace and coke-oven plant of moderate size, calculations are appended, which were based on the following assumptions:

Two isolated blast furnaces, of 350 or 450 tons daily capacity each, are installed near a large city, together with a coke-oven plant of just sufficient size to furnish the necessary coke for these blast furnaces. Coking coals of proper quality for making good blast-furnace coke can be delivered for $3 per long ton f.o.b. the furnace plant. The by-product coke is charged to the blast furnaces at $4.25 per net ton, which price is about $1.25 below the cost of standard beehive coke delivered in that locality. Coke-oven gas of a heating value of 550 B.t.u. per cubic foot, but not subject to minimum-candle-power specifications, can be used in the city for domestic purposes and would bring a price of 12c. per 1,000 cu. ft. The prevailing cost of electric power is 1.2c. per kilowatt-hour distributed. With a practically unlimited power market, the total surplus from the blast-furnace and coke-oven gases of this plant can be absorbed by the established electric power company, which operates steam turbine power stations of approximately 300,000,000 kw-hr. total output per year generated at a yearly use factor of about 38 per cent.

With such favorable local conditions, the total quantity of coke-oven gas can be used for domestic or power purposes, or both. To realize this, gas from some outside source must be used for under-firing at the coke ovens. Blast-furnace gas should not be used for this purpose in this instance, since this ideal gas-engine fuel can much more profitably be utilized for the production of electric power. Producer gas made from small coke and coke breeze in revolving grate or slagging type gas producers would answer splendidly and should be used if sufficient quantities of these fuels are available. As a general rule, however, the amount of coke breeze does not exceed 7 per cent. to 8 per cent. of the total coke made, and this quantity is not sufficient to produce the necessary gas for under-firing.

In this discussion, it was assumed that a mixture of raw coal and coke breeze is gasified in some type of mechanically-stirred gas producer equipped with apparatus for the recovery of by-products. This producer

plant would be in operation at practically full output at all times, making this method of providing for under-firing very cheap and attractive. The question whether the total quantity of coke-oven gas produced should be marketed for community use, or only its rich portion, while the lean portion is used for power purposes in the natural state, or in admixture with blast-furnace gas, can be properly answered only when the quality of the coking coals is known and when the value of the coke-oven gas is correctly appraised. This value depends entirely upon the use to which the gas is put. It is determined from case to case by the price of other fuels which could be used for the same purposes and which this gas replaces.

Distinction must be made between the "cost of production" and the "value at consumption" of such gases, because different gases vary considerably in pyrometric effect and the heating value per cubic foot of any gas compared with that of another fuel is not necessarily a criterion of its intrinsic value. For example, if blast-furnace gas and coke-oven gas are used for power purposes, their value can be determined as follows:

A blast-furnace gas-engine power plant of about 16,000 kw. installed capacity, operating at a use factor

$$\frac{\text{Total kilowatt-hours produced per year}}{\text{Maximum continuous rating of plant} \times 8{,}760} = 36 \text{ per cent.}$$

would produce about 50,000,000 kw-hr. per year.

The cost of producing electric current at the switchboard, based upon the values given in the author's recent paper published in the Year-book for 1913 of the American Iron and Steel Institute, would be about as follows:

	Cents
Net operating expenses including gas purification per kilowatt-hour...	0.15
Fixed charges 15 per cent. on the total investment of $1,440,000 ($90 per kilowatt installed capacity).....	0.43
The total cost of power per kilowatt-hour without fuel expenses would thus be.....	0.58
If electric power distributed in the locality is worth 1.2c. per kilowatt-hour and if the value of power delivered at the switchboard of existing steam power plants is.....	0.80
(deducting 0.40c. per kilowatt-hour distribution charges) then the value of fuel per kilowatt-hour consumed in the gas engine station would be 0.80−0.58=.....	0.22
Assuming a heat consumption of 19,000 B.t.u. per kilowatt-hour (over-all thermal efficiency 18 per cent. at 36 per cent. use factor) and a heating value of blast-furnace gas of 100 B.t.u. per cubic foot, the gas consumption per kilowatt-hour would be 190 cu. ft., valued at 0.22c.	
The value (not the cost) of 1,000 cu. ft. of purified blast-furnace gas would, therefore, in this particular case be $\frac{1{,}000}{190} \times 0.22 =$	1.16
On the basis of equivalent, absolute heat units and disregarding the difference in pyrometric effect (which is of little influence in gas-engine work) the value of coke-oven gas of 550 B.t.u. per cubic foot compared with that of blast-furnace gas of 100 B.t.u. per cubic foot would be 5.5×1.16= (per M cu. ft.).....	6.4

If, however, the use factor of this power plant were materially greater, for instance 80 per cent., as it could be in case of parallel operation of the gas-engine plant with existing steam-turbine installations, then the value of coke-oven gas used in gas engines would be much higher.

At 80 per cent. use factor, a 16,000-kw. power plant would produce about 112,000,000 kw.hr. annually and the cost of generation at the switchboard would be as follows:

	Cents
Net operating expenses including gas purification, per kilowatt-hour...	0.09
Fixed charges: 15 per cent. on $1,440,000.....	0.19
Total cost per kilowatt-hour without fuel expenses.....	0.28
If the value of power at the switchboard of existing steam-turbine stations is.....	0.80
as before, and if the heat consumption of the gas engines is 16,000 B.t.u. per kilowatt-hour (over-all thermal efficiency 21.5 per cent. at 80 per cent. use factor) and the heating value of blast-furnace gas 100 B.t.u. per cubic foot, then the gas consumption per kilowatt-hour would be 160 cu. ft. valued at $0.80-0.28=$.....	0.52
The value (not the cost) of 1,000 cu. ft. of blast-furnace gas would now be $\frac{1,000}{600}\times 0.52=$.....	3.25
and that of coke-oven gas on the basis of equivalent heat units 5.5 $\times 3.25$ or approximately (per M cu. ft.).....	18.0

The cost of sulphur purification of the coke-oven gas must be deducted, since this expenditure reduces the profit realizable from its utilization. But, since an equal amount would have to be subtracted from the selling price of illuminating gas, because sulphur cleaning is just as essential in one case as in the other, the values found above can be directly compared with the price obtainable by the sale of this gas for illuminating purposes.

It will be seen that the intrinsic value of these gases depends greatly upon the use factor of the power plant, and that, at a price of 12c. per 1,000 cu. ft. for domestic gas, coke-oven gas would really be more valuable as a gas-engine fuel if the power plant operates at 80 per cent. load factor throughout the year.

From this calculation the deduction could be made that, in case of an unlimited market for electric power and with nearly full output of the gas-engine station, it would be more profitable to operate gas engines on coke-oven gas than to sell the latter for domestic purposes. This conclusion would be perfectly justified if coke-oven gas could be used in very large gas-engine units just as satisfactorily as blast-furnace gas. In spite of the present remarkable development of the gas engine, such is, however, not quite the case. Especially in this country, there are but very few coke-oven gas engines of medium size in operation and these were built a number of years ago when even the blast-furnace gas engine in this country was, as it were, in its infancy.

The blast-furnace gas engine has to-day attained a very high standard, as can readily be seen by the truly remarkable results achieved with the

world's largest gas-engine plant at Gary; but the coke-oven gas engine, although mechanically equivalent to the blast-furnace gas engine, has to deal with a fuel of entirely different character.

Blast-furnace gas contains on an average not more than $3\frac{1}{2}$ per cent. hydrogen, from 25 per cent. to 28 per cent. carbon monoxide, and has but a fraction of 1 per cent. of methane; its heating value is about 95 to 100 B.t.u. per cubic foot. Coke-oven gas, on the other hand, varies greatly in composition according to the kind of coal carbonized and the time of generation during the process of carbonization. It contains from 50 per cent. to 60 per cent. hydrogen, 4 per cent. to 8 per cent. carbon monoxide, 2 per cent. to 5 per cent. illuminants and from 28 per cent. to 35 per cent. methane and its heating value ranges from 450 to 650 B.t.u. per cubic foot. Blast-furnace gas requires approximately its own volume of air, while coke-oven gas must be mixed with from 8 to 15 volumes of air for complete combustion and satisfactory operation in gas engines. The large percentage of hydrogen in coke-oven gas causes an extremely rapid flame propagation, so that the time of combustion of a mixture of coke-oven gas and air is considerably shorter, and this combustion much more of an explosive character, than is the case with blast-furnace gas. The compression in coke-oven gas engines must, on account of these characteristic qualities of the fuel, be kept considerably lower than is customary and appropriate for blast-furnace gas engines.

The principal reason for the early difficulty with coke-oven gas engines abroad was that high compression pressures were used, which resulted in excessive explosion pressures. The latter not only caused serious damage to gas cylinders, pistons, and piston rods, but were responsible for extremely unsatisfactory engine operation on account of severe premature explosions, which occurred particularly at heavy loads. Costly experience has taught European gas-engine manufacturers to reduce the compression pressure in coke-oven gas engines to 110, and even to 85 lb. per square inch, depending somewhat on the percentage of hydrogen in the gas. While, according to thermodynamic laws, a lower compression pressure is necessarily followed by a lower thermal efficiency, it has been found in practice in Germany that the actual heat consumption per b.h.p. hour of coke-oven gas engines at full load is practically the same as that of blast-furnace gas engines for the same load condition. Theory and practice thus seemingly disagree, but the explanation of this phenomenon is not difficult: one reason is the better mechanical efficiency at lower compression, but the principal reason is that coke-oven gas engines must, for practical considerations, be operated on very lean gas mixtures, that is, with a large air excess. The presence of a much larger quantity of air than is theoretically necessary, and the rapidity of flame propagation, result in perfect combustion of the charge at practically constant volume,

while blast-furnace gas-engine indicator cards frequently reveal after-burning of the mixture during expansion.

Owing to the snappy action of the gas, coke-oven gas engines are at times quite susceptible to back-firing and premature explosions. Effective cooling of gas cylinders, pistons, piston rods, etc., becomes, therefore, a very important matter on account of the intense heating of these parts. The absolute quantities of coke-oven gas consumed during one working cycle are comparatively small, even at full load, and perfect regulation of coke-oven gas engines is, therefore, not as easily accomplished as that of blast-furnace gas engines. Serious difficulties were encountered in the earlier days because it was found quite impossible to maintain satisfactory parallel operation, especially at fractional loads, of coke-oven gas-engine generators feeding into the same system. Ignorance on the designer's part was very largely responsible for these difficulties, because gas ports and gas-valve areas were made far too large, and entirely out of proportion to the size of the air ports, since no attention was paid to the low specific gravity of coke-oven gas. Much larger quantities of gas than actually required at any specific load were thus caused to enter the gas cylinders, and these earlier engines nearly always operated on excessively rich mixtures. The obvious result was, after burning, destruction of exhaust valves, piston rods, and cylinders, violent back-fires and vicious premature explosions, all of which caused extremely unsatisfactory commercial operation.

These difficulties can be readily overcome at present by correct application of a few simple laws and experiences on the designer's part. To-day the coke-oven gas engine has reached such a high state of perfection in Germany that it is second only to the blast-furnace gas engine in importance and serviceableness, with the qualification, however, that unit capacity and more particularly the rating must be conservative.

Several years ago it was thought perfectly admissible to rate coke-oven gas engines a good deal higher than blast-furnace gas engines, and it was reasoned that the higher heating value of coke-oven gas should allow of higher mean effective pressures in the gas cylinders. When it was found, however, that, after a comparatively short time of operation, these engines began to show distinct signs of distress, and caused an appalling amount of difficulties and expense, designers and operators wisely reduced the maximum capacity by strictly limiting the mean effective pressure to about 65 lb. per square inch. When it was clearly realized that coke-oven gas engines must under no circumstances be rated higher than blast-furnace gas engines, that the compression pressure should be lowered and lean mixtures must be used, the operation of plants which formerly were distinguished by their endless troubles, improved as soon as the necessary changes had been made. Several large coke-oven gas-engine stations in

Germany have now been in very successful, continuous operation for over five years.

The latest coke-oven gas engines built by Thyssen & Co., of Muelheim-Ruhr, Germany, were installed at the Bonifacius colliery, in Westphalia. Each engine is of the twin tandem double-acting four-cycle type having four gas cylinders 47½ in. in diameter, 51 in. stroke, and develops at 94 rev. per minute about 5,000 b.h.p. Alternating current of 5,000 volts is generated and used at the various pits for operating electric hoists and other machinery. The power station can be operated in parallel with the R.W.E. in Essen, and is tied in with the blast-furnace gas-engine station at Gelsenkirchen and with several steam-turbine plants in the vicinity. Parallel operation is beyond criticism and the load factor of these engines is very high—from 90 per cent. to 100 per cent.—since peak loads are taken up by the blast-furnace gas-engine plant and the steam turbines. These engines were built two years ago and are probably the largest coke-oven gas engines in the world. Previous experiences with smaller gas engines were carefully considered when these engines were designed. The compression pressure is only 85 lb. and the maximum explosion pressure does not exceed 250 lb. per square inch. Each cylinder end is provided with three igniters. The gas consumption as measured by means of a large gas holder was found to be 12,000 B.t.u. per kilowatt-hour at 90 to 100 per cent. load. The over-all mechanical efficiency, including generator, is 82.3 per cent. at that load. The engines are equipped with butterfly valve gear, and regulation is performed on the combination "quality-quantity" principle. In the opinion of Thyssen & Co., even larger coke-oven gas engines than these could be built without difficulty. The repair costs of such engines would not be any higher than those for blast-furnace gas engines, provided they receive equally efficient care.

The Friedrich-Wilhelms-Huette of Muelheim-Ruhr, Germany, have equipped with their gas engines what they contend is the largest coke-oven gas-engine plant in the world. The power house at the Prinz Regent colliery has a total capacity of 21,600 b.h.p. and contains four d. a. twin tandem gas engines of 4,000 b.h.p. each, one d. a. single tandem unit of 2,000 b.h.p. and two d. a. single tandem units of 1,800 b.h.p. This firm is now building an air compressor direct coupled with a two-cylinder twin coke-oven gas engine of 1,700 i.h.p. capacity, furnishing 8,750 cu. ft. of free air per minute compressed to 85 lb. per square inch, which is used for various purposes about the coal mines.

Ehrhardt & Sehmer, Saarbruecken, Germany, are among the foremost builders of gas engines in Europe. The largest coke-oven gas engines built by this firm have four cylinders 47½ in. in diameter, 45¼ in. stroke, and operate at 108 rev. per minute, giving 4,600 b.h.p. This firm claims to be prepared to build such engines in unit sizes of

8,000, and even 9,000 b.h.p., if such large engines were demanded, and now has under construction a unit 51 in. in diameter, 51 in. stroke, to operate on "Mond" producer gas, which in the mixture unit contains more hydrogen than coke-oven gas.

M. A. N. of Nuernberg, Germany, who were the first gas-engine builders to introduce the double-acting four-cycle gas engine in its present form, claim that they are building units of 6,000 b.h.p. capacity in four cylinders. This concern has probably the largest number of gas engines in operation throughout the world.

There are not many coke-oven gas engines of such large unit capacity in operation in Europe, and it seems that about 1,100 to 1,200 b.h.p. per double-acting gas cylinder is at present the upper limit of such engines abroad.

The overwhelming majority of gas-engine manufacturers in this country restrict the capacity of coke-oven gas engines to about 700 or 850 b.h.p. per double-acting cylinder at present and until more experience is available, which will undoubtedly be the basis for a gradual development considerably beyond the present maximum unit size. This prudent and conservative policy on the part of American gas-engine builders shows excellent judgment, and deserves warm appreciation and cordial support, because it will surely forestall troubles and disappointments. The coke-producing industries should encourage this spirit and assist the manufacturers by installing coke-oven gas engines and developing a field which has lain idle altogether too long. Ultimately the producer of electric power, using coke-oven gas engines, will be the beneficiary if he is the benefactor now.

The questions of design and material for gas cylinders and of the best system of regulation and valve gear arrangement were very thoroughly and ably discussed in a paper presented by Prof. Paul Langer of Aachen, Germany, before the recent annual meeting of the American Society of Mechanical Engineers. Professor Langer's conclusions are based on both theoretical and practical considerations and coincide with the present views of the majority of gas-engine manufacturers abroad. Careful study of this paper will prove highly interesting and instructive to those concerned in the subject. Briefly, Professor Langer advocates the continued use of cast-iron gas cylinders, preferably in one piece, and he recommends the so-called combination, "quality-quantity" regulation method, which is so easily and perfectly secured by the universal use of simple butterfly valves for gas and air control.

Turning now from the mechanical features of the coke-oven gas engine to the fuel itself, it should be remembered that the presence of sulphur in coke-oven gas in the form of sulphuretted hydrogen and carbon bisulphide was the cause of very serious troubles before it was learned how to combat its influence. The slightest water leak in the gas cylinders

from defective pistons or water-cooled exhaust valves resulted in serious corrosion of all finished parts, and particularly of piston rings, piston rods, and metallic packing. This corrosion was caused by sulphuric acid formed by the combination with oxygen and water of sulphur dioxide, resulting from the combustion of sulphur in the gas. Instead of striking at the root of the evil by proper sulphur purification, and by prevention of water leaks, attempt was made to cure the trouble by resorting to building materials which would not be attacked by sulphuric acid. The author remembers that the first single-acting coke-oven gas engine designed by him in 1901 was equipped with gas-cylinder liners and exhaust valves made of a special phosphor bronze.

Later, several gas-engine builders in Germany abandoned the use of water-cooled exhaust valves and substituted solid, non-cooled valves, in order to reduce the liability of water leaks. This practice is now well established for blast-furnace gas engines, but the most prominent and experienced manufacturers prefer to use water-cooled Durana metal exhaust valves for coke-oven gas engines on account of the much higher temperature of the exhaust gases. The true remedy was found, however, when it was learned to design pistons, cylinder heads, and gas cylinders in such a manner that cracking of these parts was avoided. But, even now, it is one of the most important duties of the operators to watch carefully for water leaks and, fortunately, the presence of water in the gas cylinders can easily be discovered by the dull black appearance of the piston rods.

The corroding influence of sulphuric acid formed by sulphur in the gas in the presence of water can often be seen on the exhaust piping whenever water has to be injected to muffle the noise of the exhaust. In several plants abroad aluminum exhaust pipes which are not attacked by acids were tried with very good success, but the prohibitive cost of the material makes its extensive use impracticable. Cast-iron pipes and mufflers with a 2-in. coating of concrete or lead-lined pipes are used in several plants and in a few exceptional cases the short cast-iron pipe connections between exhaust chambers and mufflers are simply replaced every second or third year.

Considerable progress has been made in recent years in designing sulphur cleaning plants which eliminate nearly all of the sulphuretted hydrogen, although no practical means has so far been discovered to separate carbon bisulphide from the gas. Gas-engine manufacturers in Europe are, however, willing to guarantee satisfactory and continuous operation of gas engines supplied with purified coke-oven gas containing as much as 1.25 grains of sulphur compounds in 1 cu. ft. The sulphur purification is accomplished by passing the coke-oven gas, after recovering the by-products, through filter boxes consisting of a number of sections filled with iron oxides or with a mixture of iron ore and certain

patented masses. A mat which is much used in Belgium with considerable success consists of two-thirds bog iron ore and one-third patented "Lux" mass which has the following composition:

Fe_2O_3	72.40
Al_2O_3	8.00
SiO_2	2.60
CaO	1.40
Na_2O	2.90
Na_2CO_3	5.00
H_2O	8.13
	100.43

The mat is regenerated from time to time by removal from the purifiers after a certain length of service and exposure to the atmosphere, or else continuously by adding to the gas 2 to 3 per cent. of air, which is drawn in at a convenient point between the coke-oven batteries and exhausters.

Nitric acid formed in the gas cylinders during combustion of the charge has a corroding effect upon the finished engine parts similar to that of sulphuric acid. Nitric acid is formed from the traces of ammonia remaining in the gas, after passing through the by-product plant. The danger from nitric acid is minimized if the direct ammonia process is used, since, by the latter, the ammonia is recovered from the gas so completely that hardly any trace remains.

Naphthalene, which is deposited by the gas particularly in cold weather, easily obstructs pipe lines and valves. This also has been the cause of much annoyance. If benzol is recovered from the gas no trouble from naphthalene deposits can occur, since it is absolutely necessary for efficient benzol recovery to extract the naphthalene from the gas. In plants where benzol is not recovered, it happens not infrequently that the gas pipes, etc., have to be steamed out from time to time in order to dissolve the naphthalene, which otherwise accumulates in the piping and on the valves, until the output of the power plant is noticeably reduced. It should be mentioned incidentally that benzol recovery, in connection with by-product coke-oven plants, improves gas-engine operation owing to the elimination of certain hydrocarbons, and because the heating value of the gas is reduced about 5 per cent. Back-firing and premature explosions occur much less frequently when debenzolized coke-oven gas is used in gas engines, as practical experience in a large coke-oven gas-engine plant in Germany has shown. Coke-oven gas used for power purposes, or as fuel in a steel plant, does not require any illuminating properties; the recovery of the benzol from the gas for sale offers very attractive financial possibilities and yields an exceptionally handsome profit on the investment.

A straight-forward acknowledgment of the salient difficulties with earlier coke-oven gas engines coupled with a full account of the remedies which eliminated the trouble in subsequent installations can have only a beneficial effect upon the cause of the coke-oven gas engine. We learn mostly by our mistakes, but if warning is freely given and also properly heeded, the repetition of failures can and will be avoided in the future. The meager experiences had with one or two earlier coke-oven gas-engine installations in this country are, in the author's judgment, not sufficient premises to argue for or against the use of coke-oven gas for gas-engine purposes and to draw conclusions of any value. The evident hesitancy on the part of the great majority of gas-engine users in this country to adopt the coke-oven gas engine as part of their standard power equipment is based upon lack of knowledge rather than upon discouraging facts. Just as the producer-gas engine is unjustly being blamed for the shortcomings of certain gas producer installations of the past, so the reputation of the coke-oven gas engine is still suffering from the effects of former ignorance of the peculiarities of a gas-engine fuel which has many ideal qualities but requires intelligent handling.

Coke-oven gas engines are now used abroad to no small extent, but their application has so far been more or less limited to the generation of electric power. The principal reason is that European, and particularly German, coke-oven plants are usually located at the coal mines, where no demand for other forms of power exists. In a few instances coke-oven gas blowing engines share in the blowing of blast furnaces, but only with a view to insuring the safety of the furnace plant in case the supply of air delivered by the blast-furnace gas blowing engines should, for some reason, suddenly fail. In these very rare cases, the coke-oven gas engine is merely a rather unnecessary factor of safety, which can readily be dispensed with in modern blast-furnace plants. The application of the coke-oven gas engine for direct driving of rolling mills is to the author's knowledge not illustrated by a single installation abroad or in this country. No inherent difficulty peculiar to the coke-oven gas engine is responsible for this condition; in fact many considerations point clearly to the advisability of its use for this purpose. Its absence from this field is largely explained by the geographical separation of coke-oven plants and rolling mills.

The load conditions of rod, wire, and sheet mills are not unfavorable for direct gas-engine drives, since they impose upon the prime mover neither violent nor excessive load variations. The average power demand of such mills is moderate, so that frequently single tandem gas-engine units are fully capable of performing the work. The rope wheel, which usually drives a number of stands, is necessarily of large diameter and can easily be given sufficient weight to provide for ample flywheel effect.

Uniformity of rotation is thus insured and it is not difficult to obtain a perfectly uniform product. The direct-connected rolling-mill gas engine for light continuous mills does not require much overload capacity, and can therefore be suitably rated to operate at a high load factor. The gas consumption is then very low and the thermal efficiency high. The result is a substantial fuel economy per ton of finished product, compared with steam-engine, steam-turbine, or electric-motor drives.

The principal advantage of direct gas-engine mill drives is, of course, the saving in cost of installation and operation. Such direct drives do away with costly combinations of electric central station, transformers, transmission lines, mill motors, and the requisite expensive auxiliaries. They are, moreover, not attended by the multiplicity of efficiency losses on account of the roundabout way of generation, transformation, and transmission of energy, which are handicapping electric mill drives. Blast-furnace gas engines are used for light rolling mill purposes in a number of steel plants abroad and they have proved the adaptability and serviceableness of the gas engine for driving direct-connected roll trains in years of satisfactory service.

In our country the installation of by-product coke-oven plants located near blast furnaces and steel mills is rapidly becoming distinctly American practice. The use of coke-oven gas, or of a suitable mixture of blast-furnace and coke-oven gas, in gas engines for rolling-mill purposes has, therefore, a better future here than abroad.

Blast-furnace gas of low heating value and great volume does not lend itself readily to economical long-distance transportation through pipe lines, because the first cost of compressor equipment and pipe line would alone be prohibitive. Besides there would be the high maintenance and operating expenses of such an arrangement. For coke-oven gas containing from five to six times the amount of heat energy in a given volume, these conditions are much more favorable. The transmission of coke-oven gas for illuminating purposes through distances of 80 and 90 miles is not unknown in this country, and the expenses incidental to the installation and operation of booster stations are moderate. Similar conditions obtain in the case of piping a mixture of blast-furnace gas and coke-oven gas, although, of course, in a less degree. In this country, large steel plants are frequently so extensive, and cover so much ground, that the distance from the blast furnaces to the light rolling mills is considerable. Sometimes these mills are operated under different management, or even by independent corporations. The use of coke-oven gas, and the installation of direct-connected gas engines for driving these mills, may, in such instances, offer both financial and operating advantages. Manifestly, no general rule can be laid down which would apply in all cases where the question of direct gas-engine drives for rolling mills is at issue. Local

conditions of load, of arrangement of mills, of distance from the source of gas supply, and others, must be carefully considered, and the intrinsic value of the gas for this particular purpose must be determined from case to case, in order to realize the greatest possible returns from its utilization, whether inside or outside of the steel plant. Each single case must, therefore, be examined on its own merits, and the influences must be intelligently weighed, which individually and collectively bear upon the financial result of such installations.

No matter what objections could justly or unjustly be raised against the use of coke-oven gas for direct power generation in modern gas engines, none of these can possibly apply to the use of a mixture of blast-furnace and coke-oven gas. The practice of combining blast furnaces, steel mills, and coke-oven plants in one locality gives the American iron and steel master an enormous advantage over the majority of his European competitors in the matter of rational utilization of the available surplus gases. It is evident what truly ideal conditions would obtain in such combined plants if only one kind of gas fuel were used for all requirements of the plant. A gas mixture in suitable proportions of blast-furnace and coke-oven gas would fulfill the most exacting specifications which could be imposed. It could be used to heat the blast for the blast furnaces, to operate blast-furnace and Bessemer gas-blowing engines, to produce electricity in gas electric central stations, to run the pumping plant, to serve as bench fuel at the coke ovens, to heat the pig-iron mixers, to produce steel in open-hearth furnaces, to heat the ingots in the soaking pits, and the steel in the reheating furnaces, to drive gas engines direct connected to certain roll trains, to supply the needs of the foundry, and, finally, to raise steam in boilers. Such a gas mixture would offer these steel plants even greater advantages than those enjoyed by a few others in this country which are still blessed with an adequate supply of natural gas.

The scope of this paper does not permit discussion of the metallurgical uses of gas fuels, and reference should be had to Emile Houbaer's recent paper on this subject which was read before the (British) Iron and Steel Institute's October meeting at Brussels. Houbaer's exceedingly valuable contribution to the knowledge of the application of blast-furnace, coke-oven, and producer gas for steel-making purposes illustrates particularly recent achievements at the works of the John Cockerill Co., of Seraing, Belgium, which has truly been the pioneer in the economic utilization of industrial surplus gases in its different phases.

It was pointed out in previous paragraphs that both blast-furnace gas and coke-oven gas possess certain characteristic qualities which are advantageous or disadvantageous from the standpoint of gas-engine operation. The use of a mixture of these gases very happily combines the good points of both, while it eliminates nearly all their objectionable

features. The addition of a small portion of coke-oven gas to a much larger quantity of blast-furnace gas results in a moderate increase of both heating value and ignition velocity of the gas-engine fuel mixture. With a suitable compression pressure perfectly satisfactory results from the use of such mixtures in gas engines are assured. The amounts of hydrogen and hydrocarbons are materially smaller than in coke-oven gas, but somewhat greater than in blast-furnace gas, so that this mixture in composition and effect is somewhat similar to air-blown producer gas. A segregation of hydrogen will not occur if the mixing of the constituent gases is thorough, and if the gas holder is designed to prevent pocketing and stagnation of the gas mixture, which is not a difficult matter. The first requirement can be very easily accomplished by scrubbing both gases simultaneously in regular static and dynamic gas washers, which are necessary for cleaning blast-furnace gas for engine purposes. The so-called secondary gas washers of the centrifugal type, whether fans, disintegrators, or Theisen washers, produce such a thorough and uniform mixture of the gases that segregation of the elements of low specific gravity cannot occur in the short time of travel of the gas mixture from the gas-washing plant to the place of consumption.

There would not be any difficulty in maintaining a heating value of the gas mixture to all intents and purposes uniform, if this task were committed to the action of automatic devices. One, or, if necessary, several, automatic recording gas calorimeters—for instance, of the "Smith" type—could be equipped with a maximum-minimum electric-contact attachment, corresponding to the permissible maximum and minimum heating value of the gas mixture. These calorimeters would control—by means of relays and electric motors—gate or butterfly valves, arranged in the coke-oven gas pipe at, or near, the point of connection with the blast-furnace gas main.

The suggestion of using a gas mixture for gas-engine purposes is by no means new or original, with the exception perhaps of the proposed method of controlling the heating value automatically. Gas-engine plants using such mixtures have been in very successful operation abroad for a number of years. Owing to intelligent design and to simple butterfly valve gear regulation, variations of considerable magnitude in the heating value of the gas mixture are apparently without detrimental effect upon the operation of these engines. The heating value of the mixture is controlled by hand at these plants. A mixture of one part of coke-oven gas of 500 B.t.u. and of 15 parts of blast-furnace gas of 100 B.t.u. would have a heating value of 125 B.t.u. per cubic foot.

Let us assume that the chemical composition by volume of the two original gases in some particular case is as follows:

Blast-Furnace Gas, Per Cent.		Average Coke-Oven Gas, Per Cent.	
CO	26.0	CO	7.5
CO_2	12.7	CO_2	2.0
H	3.0	H	51.0
CH_4	0.3	CH_4	29.5
$C_n H_m$	0.0	$C_n H_m$	2.0
N	58.0	N	8.0
	100.0		100.0
B.t.u.	101	B.t.u.	500

If one part of coke-oven gas is mixed with fifteen parts of blast-furnace gas, the chemical composition of the resulting mixture would be approximately the following:

1 : 15 Coke-Oven Gas : Furnace Gas Mixture	
CO	24.8 per cent.
CO_2	12.0
H	6.0
CH_4	2.1
$C_n H_m$	0.1
N	55.0
	100.0 per cent.
B.t.u.	125

The composition of this gas mixture corresponds very nearly to that of producer gas made from coke or anthracite coal, as will be seen from the low content of hydrogen and hydrocarbons, etc. A gas mixture of this typical composition would make an ideal gas-engine fuel. A gas-engine plant of much larger capacity could be installed and operated, owing to the increase in quantity, and the improvement in quality, of the available gas fuel. The argument that the admixture of valuable coke-oven gas to blast-furnace gas would be uneconomical, on account of the unavoidable losses caused by large quantities of gas escaping unused at times of light load, is justified only if the load factor of the power station is very low. This will not be the case, however, if the gas-engine station is operated in parallel with existing steam power plants. When carrying the bulk of the average load in the gas-engine station, leaving peak loads to be taken care of by the steam turbo-generators, these gas losses would be of very little importance.

If such arrangements are made with the public service corporation operating steam power plants in the district, the largest possible gas-engine power station should be installed, limited in capacity only by the average constant hourly power demand and, of course, by the amount of available surplus gas mixture. Spare units in the gas-engine station are unnecessary, since a number of existing steam turbo-generators would stand by in case part of the gas power plant were, for any reason, temporarily disabled. The maximum capacity of the gas-engine power station should not be restricted too much by applying an excessive arbitrary "safety factor"

in determining what portion of the total available surplus gas should be depended upon for power generation. The fact that the gas mixture is supplied by two practically independent sources constitutes in itself a safety factor of no little value. At times of momentary shortage of either ingredient the regular output of electric power would be maintained by admitting to the engines a mixture leaner or richer than standard. The possibility of using in the engines, producer gas made by one of the existing blast furnaces operating according to Brassert's gas-producer principle, as mentioned elsewhere, is an adequate safeguard against the influence of a curtailment of pig iron and consequently blast-furnace gas production brought about by a business slump. Blast-furnace producer gas has about the same heating value as the gas mixture and its composition by volume is approximately as follows:

Blast-Furnace Producer Gas

CO	34.0 per cent.
CO_2	1.6
H	2.9
CH_4	1.0
B.t.u.	126

The percentage of hydrogen in furnace producer gas is considerably smaller than in the standard gas mixture. The former gas can, therefore, be substituted for the mixture with impunity. The higher percentage of carbon monoxide in the producer gas will counterbalance the reduction in hydrogen to a large extent, as far as the effect upon gas-engine operation is concerned. It can thus be expected that the change from one gas to the other can be made even without important readjustments of valve and damper setting on the gas engines.

Conditions could arise during a blast-furnace campaign which may necessitate the temporary curtailing of gas production of one or both furnaces. Delays caused by casting, changing tuyères, and minor repairs, requiring generally but a few minutes, are harmless for the gas-engine power plant if, following common practice, a gas holder of sufficient capacity is installed. As far as the question of relining the blast furnaces is concerned, this necessity occurs but once in three or four years and requires from one to two months. Both blast furnaces would of course never be relined simultaneously, so that one furnace would always be available to operate as a gas producer. The quantity of gas made by the producer furnace is ample to generate all the power which a mixture of blast-furnace gas from two iron-producing blast furnaces and of the required amount of coke-oven gas could produce in times of normal business conditions.

The proper choice of the "safety factor" of surplus gas utilization, to which allusion was made, is entirely a matter of local conditions and of

good judgment. In the case under discussion the highest possible factor of safety was chosen to prove the point in question.

Whatever the capacity of the gas-engine plant is, both gas-blowing and electric engines should be equipped with exhaust-heat boilers. Steam of any desired gauge pressure can be generated in such boilers at very small cost, and over 50 per cent. of the heat ordinarily rejected in the exhaust can be recovered. The steam can be utilized in many ways: for heating purposes, for operating steam turbo-generators, or for blowing gas producers, especially if by-product recovery gas producers are installed in connection with the coke-oven plant. Practical experience in a number of European gas-power plants shows that about 1.8 lb. of steam of 150 lb. gauge pressure can be raised in this manner for each brake horse power developed in gas engines operating at full load. This quantity increases to about 2 lb. per brake horse power if the gas engines operate at approximately half load. It will be shown later that the requirements of about 1.5 lb. of steam per pound of coal gasified in the by-product recovery gas-producer plant are fully covered by the amount of steam raised by exhaust-heat boilers in the gas-engine station contemplated in this instance.

Before giving comparative detailed calculations to prove the financial advantage of using a mixture of blast-furnace and coke-oven gas instead of straight blast-furnace gas for power generation in a gas-engine plant, a few points regarding the cost of the fuels entering into this calculation have to be discussed.

It is very difficult to determine the actual cost of generating blast-furnace gas and it is therefore customary to appraise this gas on the basis of the value in heat of an equivalent quantity of coal. In the present instance coal of 13,500 B.t.u. per pound costs \$3 per long ton and 1,000,000 B.t.u. are worth $\frac{1{,}000{,}000 \times 300}{13{,}500 \times 2{,}240} = 10$c. Blast-furnace gas from merchant furnaces having a coke consumption of about 2,400 lb. per ton of foundry iron contains about 101 B.t.u. per cubic foot; 1,000 cu. ft. of this gas are therefore worth $\frac{1{,}000 \times 101}{1{,}000{,}000} \times 10 = 1.01$c.

Coke-oven gas of 550 B.t.u. appraised on the same basis is worth $\frac{1{,}000 \times 550}{1{,}000{,}000} \times 10 = 5.5$c. per 1,000 cu. ft. It was shown that the intrinsic value of this gas varies according to the method of utilization. For high-temperature work, such as reverberatory furnaces or the like, the appreciation due to the higher pyrometric effect amounts to approximately 35 per cent., so that its value for these purposes becomes 7.5c. per 1,000 cu. ft. and the value of 1,000,000 B.t.u. increases to 13.5c. This fact is of considerable importance in the matter of selection of the most economical bench-fuel for the coke-oven plant. A little calculation will readily show the point in question:

If producer gas generated in a by-product recovery gas-producer plant were used for under-firing, there would be required $\frac{1,150}{135} = 8.5$ cu. ft. of this gas per pound of coal, assuming that 1,150 B.t.u. are necessary to carbonize 1 lb. of coal, and that the by-product producer gas has a heating value of 135 B.t.u. per cubic foot. A coke-oven plant producing 1,000 tons of coke per day at a yield of 75 per cent. of coke from coal carbonizes approximately 2,700,000 lb. of coal in 24 hr., requiring about 23,000,000 cu. ft. of by-product producer gas. Assuming that one short ton of coal generates 125,000 cu. ft. of gas, there would have to be gasified 185 short tons of coal per day. The cost of making this gas would be as follows:

1. Charges per year:

(a) Fixed charges at $1,600 investment in the complete plant, per ton of coal gasified, figuring on 6 per cent. interest, 5 per cent. amortization, and 1 per cent. for taxes and insurance, or a total of 12 per cent., =

$0.12 \times 1,600 \times 185 =$ $35,600.00

(b) Coal: $185 \times \frac{2,000}{2,240} \times 3 \times 365 =$ 180,000.00

It is assumed that the necessary steam for producer operation is generated free of charge, in exhaust-heat boilers attached to the gas engines, so that no expenditure for steam coal has to be considered.

(c) Conversion Cost: This includes all labor, materials, supplies, repairs, gas purification, acid and bags for sulphate of ammonia, etc., and amounts approximately to $1 per short ton of coal, or:

$185 \times 1 \times 365 =$ $67,500.00

The total charges per annum are, therefore, $283,100.00

2. Credits: Coal containing about 1.4 per cent. of nitrogen yields about 70 lb. of sulphate of ammonia and 7 gal. of tar per short ton of coal gasified. At a market price of $60 per ton of sulphate and 2.5c. per gallon of tar, the net returns per annum are:

(d) From sulphate of ammonia: $70 \times 185 \times 365 \times 0.03 =$ $142,000.00

From tar: $7 \times 185 \times 365 \times 0.025 =$ 11,800.00

The total yearly credits (without gas) are thus $153,800.00

The net cost of by-product producer gas is, therefore:

$283,100 - $153,800 = $129,300.00

The total quantity of coal gasified per year is $185 \times 365 = 67,500$ tons, giving about 8,500,000 M cu. ft. of gas.

The cost per 1,000 cu. ft. is therefore $\frac{129,300.00}{8,500,000}$ = approximately 1.5c.

If this gas has a heating value of 135 B.t.u. per cubic foot, the cost of 1,000,000 B.t.u. is $\frac{1,000,000}{1,000 \times 135} \times 1.5 = 11.1$c.

Comparing the value of coke-oven gas with the cost of by-product producer gas on a B.t.u. basis, it will be seen that it is a matter of economy to use producer gas for bench fuel at the coke ovens, if coke-oven gas can be utilized to better advantage for outside purposes. This fact is particularly emphasized if coke-oven gas can be sold for domestic heating and lighting at a price well above the actual cost of its manufacture.

In the following calculations use was made of the arguments discussed on the previous pages. The assumptions on which these calculations are based are stated elsewhere in the body of this paper. It should be explained that two alternatives were considered and investigated, namely, the installation of a gas-engine power plant using straight blast-furnace gas, and that of engines operating on a mixture of coke-oven gas and blast-furnace gas in the proportion 1 to 15. A further subdivision was thought necessary to account for a difference of opinion which still prevails among foundry-iron interests whether merchant furnaces of 350 tons or 450 tons capacity should be built. While the author does not feel competent to take sides in this controversy, he wishes to point out that, according to H. A. Brassert, many important considerations favor the installation of larger furnaces:

"Roughly, the cost of a plant of two 450-ton furnaces is $2,000,000 complete with gas-blowing engines, modern ore-handling machinery, bins, etc. Two 350-ton furnaces will cost very nearly the same amount, as it would not be possible to save any appreciable amount of money on the larger items such as ore-handling machinery, gas-blowing engine equipment, etc.

"The operating cost of a 450-ton furnace is almost the same per day as that of a 350-ton furnace, with the exception of the materials charged, viz., ore, coke, and limestone, which are the same per ton but more per day. The cost above materials per ton, being the same per day in both cases, would be little more than seven-ninths of that of the smaller furnaces. For instance, if the cost above materials charged were $1.50 per ton for the smaller furnaces, it would be about $1.25 for the larger, thus saving 25c. per ton of pig iron on 'cost above' alone. A saving should also be made on cost of coke per ton of iron, since a larger blast furnace can be operated more economically on fuel than a small furnace—at least such has been the experience at those plants where large furnaces of good design have been built with proper stock distribution. Large merchant furnaces making foundry iron have run on less than 2,300 lb. of coke per ton of iron, if coke of good quality is used. Smaller furnaces, such as foundry-iron interests are in the habit of building, cannot show anywhere near such practice over a period of years. Radiation losses are very much smaller on large furnaces and operation is less sensitive and more regular, old-fashioned beliefs to the contrary notwithstanding. The lowest fuel records on special grades of iron, including ferro-silicon and ferro-manganese, have been made on furnaces of the larger size."

Calculated Values

Item	Unit	2 B.F., 350 tons each	2 B.F., 450 tons each
A. Blast-Furnace Gas Used for Power Purposes			
1. Pig-iron production	24 hr.	700 tons	900 tons
2. Do	Year	230,000 tons	300,000 tons
3. Coke consumption	Ton pig	2,400 lb.	2,400 lb.
4. Rate of B.F. gas production	Ton pig	170,000 cu. ft.	170,000 cu. ft.
5. Heating value of B.F. gas	Cu. ft.	101 B.t.u.	101 Btu.
6. Total quantity B.F. gas produced	24 hr.	120,000,000 cu. ft.	153,000,000 cu. ft.
7. Required for stoves, lost by leakage (40 per cent. total)	24 hr.	50,000,000 cu. ft.	63,000,000 cu. ft.
8. Remainder	24 hr.	70,000,000 cu. ft.	90,000,000 cu. ft.
9. Gas-blowing engine requirements. a. Normal blast pressure 15 lb. per sq. in.			
10. Gas : air ratio		1.39	1.39
11. Air required at tuyères $\frac{\text{Item 4} \times \text{Item 1}}{1{,}440 \times \text{Item 10}}$	Min.	60,000 cu. ft.	77,000 cu. ft.
12. Actual air required at Dyblie valves (5 per cent. added for blowing down stoves, leaky pipes, hot-blast valves, etc.)	Min.	63,000 cu. ft.	80,800 cu. ft.
13. Vol. eff. of blowing tubs at 15 lb. blast pressure		90 per cent.	90 per cent.
14. Necessary displacement of tubs for both furnaces	Min.	70,000 cu. ft.	90,000 cu. ft.
15. Work on adiabatic compression to 15 lb. (5 BHP per 100 cu. ft.; MEP = 11.5 lb.)	Hr.	3,500 BHP.	4,500 BHP.
16. IHP required in blowing cyls. (compression eff. 95 per cent.)	Hr.	3,680 HP.	4,740 HP.
17. BHP required at gas-engine shaft (mech. eff. of blowing tubs 95 per cent.)	Hr.	3,900 BHP.	5,000 BHP.
18. IHP required in gas cylinders (mech. eff. of gas eng. 80 per cent.)	Hr.	4,900 IHP.	6,200 IHP.
19. Type of gas-blowing engines	Each	Single tandem 4 cycle double acting.	Single tandem 4 cycle double acting.
20. Principal dimensions	Each eng.	44 and 76×60 in.	44 and 76×60 in.
21. No. of gas-blow. eng. installed		5	5
22. No. of gas-blow. eng. operating		4	4
23. Net air piston area (15-in. rod)	Each tub	4,360 sq. in.	4,360 sq. in.
24. Net gas piston area (15-in. rod)	Each cyl.	1,344 sq. in.	1,344 sq. in.
25. Ratio of net areas : air : gas		3.25	3.25
26. Air-tub piston displacement per revolution	Each eng.	300 cu. ft.	300 cu. ft.
27. Speed required for full displacement	Each eng.	58 RPM.	75 RPM
28. IHP maximum continuous rating at 71 lb. MEP	Each eng.	1,650 IHP.	2,170 IHP.
29. BHP max. cont. rating at 71 lb. MEP (80 per cent. mech. eff.)	Each eng.	1,320 BHP.	1,740 BHP.
30. Gas-blowing engine requirements. b. Max. blast press. 25 lb. per sq. in.			
31. (MEP adiabatic compression = 17 lb. per sq. in.)			
32. BHP reqd. at gas eng. shaft	4 eng.	5,720 BHP	7,360 BHP
33. IHP reqd. in gas cylinders	4 eng.	7,120 IHP	9,200 IHP
34. MEP reqd. in gas cylinders	Each eng.	77 lb. per sq. in.	76 lb. per sq. in.
35. Gas consumption of gas-blow'g engines:			
36. Yearly average thermal eff. referred to BHP and 15 lb. aver. Blast pressure including (0.98) gas-cleaning efficiency		20 per cent.	20 per cent.
37. Heat consumption (15 lb. aver. blast pressure)	BHPH	12,700 B t.u.	12,700 B.t.u

38. Gas consumption (101 B.t.u. gas)	BHPH	125 cu. ft.	125 cu. ft.
39. Total gas consumption (yearly average)	Hr.	485,000 cu. ft.	625,000 cu. ft.
40. Do.....	24 hr.	11,700,000 cu. ft.	15,000,000 cu. ft.
41. Requirements of blast-furnace auxiliaries (pumps, air compressors for mud guns, bells, etc., electric light and power for ore unload , scale and trans. cars, skip hoists, pig mach., etc., power produced in gas elect. engines).....	Hr.	60,000 cu. ft.	80,000 cu. ft.
42. Gas reqd. for auxiliaries.....	24 hr.	1,440,000 cu. ft.	1,920,000 cu. ft.
43. Surplus gas available for outside purposes (Item 8—Item 40—Item 42).....	24 hr	56,860,000 cu. ft.	73,080,000 cu. ft.
44 Do.....	Hr.	2,360,000 cu. ft.	3,050,000 cu. ft.
45. B.t.u. available (101 B.t.u. gas).....	Hr.	240,000,000 B.t.u.	308,000,000 B.t.u.
46. Theoretical max. surplus elec. power obtainable (full load: 23 per cent. therm. eff.; 15,000 B.t.u. per kw-hr.)	Hr.	16,000 kw.	20,500 kw.
47. Do.....	Ton pig	23 kw.	23 kw.
48. Unit size of gas engines installed.....	Each	TT 44 in. × 60 in. 83.3 RPM: Max. Cont. Rating 2,700 kw. (71 lb. MEP): overload capacity for 2 hr. 3,000 kw. (79 lb. MEP)	

NB. Gas-Engine Plant Operating Jointly with Existing Steam-Turbine Stations

49. "Factor of Safety" acct. shorter periods of gas deficiency occasioned by casting, checking, changing tuyères and smaller repairs on equipment.....	.....	85 per cent.	80 per cent.
50. Practical max. surplus electric power obtainable.....	Hr.	13,500 kw.	16,500 kw.
51. Do (approx.)	Ton pig	19 kw.	18 kw.
52. No. of units in operation.....	.....	5	6
53. No. of spare units (for internal service)	.....	1	1
54. Total number of units installed.....	.....	6	7
55. Max. cont. rating *all* units	Hr.	16,000 kw.	19,000 kw.
56. Max. cont. rating *operating* units.....	Hr.	13,500 kw.	16,000 kw.
57. Steam of 150 lb. gauge pressure recovered by exhaust-heat boilers at 70 per cent. to 80 per cent. average load 1.85 lb. per BHPH = 2.65 lb. per kw-hr.			
58. Do. from gas-blowing engines. Item 17 × 1.85 (approx.).....	Hr.	7,000 lb.	9,000 lb.
59. Do. from gas electric engines. 0.85 × Item 56 × 2.65 (approx.).....	Hr.	30,000 lb.	36,000 lb.
60. Total average steam available.....	Hr.	37,000 lb.	45,000 lb.
61. Average possible output turbo generators at 15.5 lb. per kw-hr.	Hr.	2,400 kw.	2,900 kw.
62. No. and rating (50° C. temperature rise) possible turbo generators of about 20 per cent. overload capacity.....	.....	1–2,000 kw.	1–2,500 kw.
63. Per cent. steam capacity: gas capacity. Item 61 : Item 56.....	.....	18 per cent.	18 per cent.
64. Possible max. cont. rating power plant (gas and steam) *all* units. Item 55 plus Item 62.....	Hr.	18,000 kw.	21,500 kw.
65. Do. (approximately).....	Ton pig.	25 kw.	24 kw.
66. Kw-hr. produced in gas elec. station. Item 56 × 8,760 × Item 67.	Year	100,000,000 kw-hr.	120,000,000 kw-hr.
67. Use factor of power plant. $\frac{\text{Item 66}}{\text{Item 56} \times 8{,}760}$	Yearly	85 per cent	86 per cent

68. Gas consumption of gas electric engines: Thermal eff. 21.5 per cent. 16,000 B.t.u. = 158 cu. ft. per kw-hr. 158 × Item 66	Year.	$15{,}800 \times 10^6$ cu. ft.	$19{,}000 \times 10^6$ cu. ft.
69. Aver. gas consumption gas electric engines (1 yr. = 365 days)	24 hr.	43,000,000 cu. ft.	52,000,000 cu. ft.
70. Actual gas surplus available for power purposes. Item 43	24 hr.	56,860,000 cu. ft.	73,080,000 cu. ft.
71. Aver. utilization factor of power gas. Item 69 : Item 70	.....	76 per cent.	71 per cent.

B. Mixture of Blast-Furnace and Coke-Oven Gas Used for all Power Purposes

(Items 1 to 37 inclusive apply equally on this alternative)

1 cu. ft. of coke-oven gas of 500 B.t.u. per cu. ft. plus 15 cu. ft. of blast-furnace gas of 101 B.t.u. per cu. ft. = 16 cu. ft. of mixture of 125 B.t.u. per cu. ft.

72. Gas consumption of gas-blowing engines. $\frac{12{,}700 \text{ B.t.u.}}{125 \text{ B.t.u./cu. ft.}} =$	BHPH	102 cu. ft.	102 cu. ft.
73. Total gas consumed (yearly av.) by gas-blowing engines	Hr.	400,000 cu. ft.	510,000 cu. ft.
74. Do.	24 hr.	9,600,000 cu. ft.	12,200,000 cu. ft.
75. Requirements of blast-furnace auxiliaries. (See Item 41.)	Hr.	48,500 cu. ft.	65,000 cu. ft.
76. Gas mixture reqd. for blast-furnace auxiliaries	24 hr.	1,170,000 cu. ft.	1,560,000 cu. ft.
77. Blast-furnace gas remainder (Item 8)	24 hr.	70,000,000 cu. ft.	90,000,000 cu. ft.
78. Coke-oven gas added for 1 : 15 mixture.	24 hr.	4,650,000 cu. ft.	6,000,000 cu. ft.
79. Total gas mixture supply	24 hr.	74,650,000 cu. ft.	96,000,000 cu. ft.
80. Surplus gas available for electric power. Items 79-74-76	24 hr.	63,880,000 cu. ft.	82,240,000 cu. ft.
81. Do. (yearly average)	Hr.	2,650,000 cu. ft.	3,425,000 cu. ft.
82. B.t.u. available (125 B.t.u. gas mixture).	Hr.	330,000,000 B.t.u.	430,000,000 B.t.u.
83. Theoretical max. surplus electric power obtainable (full load; 23 per cent. thermal eff.; 15,000 B.t.u. per kw-hr.)	Hr.	22,000 kw.	28,600 kw.
84. Do. (approximately)	Ton pig.	31 kw.	32 kw.
85. Unit size of gas engines installed. (See Item 48.)	Each	TT 44 in. × 60 in. 83.3 RPM.: Max. cont. rating 2,700 kw. (71 lb. MEP). Overload capacity for 2 hr. 3,000 kw. (79 lb. MEP).	

NB. Gas-Engine Plant Operating Jointly with Existing Steam-Turbine Stations

86. "Factor of Safety" (for definition see Item 49)	.....	86 per cent.	85 per cent.
87. Practical max. surplus electric power obtainable	Hr.	19,000 kw.	24,300 kw.
88. Pract. max. surplus elect. power obtainable (approx.)	Ton pig	27 kw.	27 kw.
89. No. of units in operation	.....	7	9
90. No. of spare units (for internal service).	.....	1	1
91. Total No. units installed	.....	8	10
92. Max. cont. rating *all* units	Hr.	21,600 kw.	27,000 kw.
93. Do. *operating* units	Hr.	19,000 kw.	24,300 kw.
94. Steam of 150 lb. gauge pressure recovered by exhaust-heat boilers: at 70 per cent. to 80 per cent. average load 1.85 lb. per BHPH = 2.65 lb. per kw-hr.			
95. Do. from gas-blowing engines. Item 17 × 1.85 (approx.)	Hr.	7,000 lb.	9,000 lb.
96. Do. from gas electric engines. 0.85 × Item 93 × 2.65 (approx.)	Hr.	43,000 lb.	55,000 lb.
97. Total aver. steam available	Hr.	50,000 lb.	64,000 lb.

98.	Aver. possible output turbo generators at 15.5 lb. per kw-hr	Hr.	3,200 kw.	4,100 kw.
99.	No. and rating (50° C. temperature rise) possible turbo generators of about 20 per cent. overload capacity	.....	1–2,500 kw.	1–3,500 kw.
100.	Per cent. steam capacity: gas capacity. Item 98 : Item 93	.....	17 per cent.	17 per cent.
101.	Possible max. cont. rating power plant (gas and steam) *all* units, Item 92 plus Item 99 (approx.)	Hr.	24,000 kw.	30,500 kw.
102.	Do. (approximately)	Ton pig	34 kw.	34 kw.
103.	Kw-hr. produced in gas electric station. Item 93 × 8,760 × Item 104	Year	140,000,000 kw.	180,000,000 kw.
104.	Use factor of power plant $\frac{\text{Item 103}}{\text{Item 93} \times 8{,}760}$	Yearly	85 per cent.	85 per cent.
105.	Gas consumption of gas electric engines: Thermal eff. 21.5 per cent.; 16,000 B.t.u. = 128 cu. ft. per kw.-hr. 128 × Item 103	Year	$18{,}000 \times 10^6$ cu. ft.	$23{,}000 \times 10^6$ cu. ft.
106.	Aver. gas consumption gas electric engines (1 yr. = 365 days)	24 hr.	49,500,000 cu. ft.	63,000,000 cu. ft.
107.	Actual gas surplus available for power purposes. Item 80	24 hr.	63,880,000 cu. ft.	82,240,000 cu. ft.
108.	Average utilization factor of power gas. Item 106 : item 107	.....	77 per cent.	77 per cent.

Coke-Oven Plant

109.	Coke required per day for blast furnaces. Item 1 × Item 3	24 hr.	1,680,000 lb.	2,160,000 lb.
110.	Do.	24 hr.	840 short T.	1,080 short T.
111.	Coal required at 75 per cent. yield $\frac{\text{Item 110}}{0.75}$	24 hr.	1,120 short T.	1,440 short T.
112.	Do. with allowance of 6 per cent. coal for coke breeze	24 hr.	1,230 short T.	1,570 short T.
113.	Number of $13\frac{1}{2}$ ton ovens 18 hr. coking time. $\frac{\text{Item 112} \times 18}{13.25 \times 24} =$	.....	70	89
114.	Total coke-oven gas produced @11,000 cu. ft. per short ton of coal	24 hr.	13,500,000 cu. ft.	17,250,000 cu. ft.
115.	Coke-oven gas not required for power purposes (account utilization factor, Item 108) and therefore available for sale: $\frac{1}{16} \times$ (Item 107—Item 106) yearly aver. approximately	24 hr.	900,000 cu. ft.	1,200,000 cu. ft.
116.	Grand total coke-oven gas available for sale: Item 114—Item 78 plus Item aver. 115	24 hr.	9,750,000 cu. ft.	12,450,000 cu. ft.
117.	Percentage of total coke-oven gas used for power purposes: $\frac{\text{Item 78—Item 115}}{\text{Item 114}} \times 100$ approx.	.....	28 per cent.	28 per cent.
118.	Percentage of total coke-oven gas used for sale for domestic heating and lighting. $\frac{\text{Item 116}}{\text{Item 114}} \times 100$	.....	72 per cent.	72 per cent.

Cost of Producing Coke-Oven Gas

119.	Total investment in coking plant	.....	$1,700,000	$2,000,000
120.	Fixed charges: 6 per cent. interest plus 5 per cent. amortization plus 1 per cent. insurance and taxes = 12 per cent., $\frac{\text{Item 119} \times 0.12.}{365}$	24 hr.	$560.00	$660.00

121.	Coking coal @ $3.00 per long ton = $2.67 per short ton. Item 112×2.67..	24 hr.	$3,280.00	$4,190.00
122.	Conversion cost including all labor, material, repairs, acid, lime, bags for sulphate, etc.....	Short T. coal	75c.	72c.
123.	Do. $\frac{\text{Item 112}\times\text{Item 122}}{100}$	24 hr.	$920.00	$1,130.00
124.	Cost of by-product producer gas for heating coke ovens. (See p. 74) approximately	24 hr.	$330.00	$420.00
125.	Cost of sulphur purification of coke-oven gas @ $0.006 per M. cu. ft. Item $\frac{114\times0.006}{1,000}$	24 hr.	$80.00	$100.00
126.	Total charges	24 hr.	$5,170.00	$6,500.00
127.	Credit for sulphate of ammonia @ 22 lb. per ton of coal @ $60 per ton of sulphate. $\frac{\text{Item 112}\times22\times60}{2,000}$.....	24 hr.	$810.00	$1,040.00
128.	Credit for tar @ 6 gal. per ton @ 2¼ c. per gal. Item 112×6×0.025.....	24 hr.	$190.00	$230.00
129.	Value of coke as charged to blast furnaces @ $4.25 per short ton. Item 110 ×4.25 = ...	24 hr.	$3,570.00	$5,490.00
130.	Total credits	24 hr.	$4,570.00	$5,860.00
131.	Net cost of total coke-oven gas. Item 130—Item 126.....	24 hr.	$600.00	$640.00
132.	Net cost of total coke-oven gas delivered into holder. $\frac{\text{Item 131}\times100\times1,000}{\text{Item 114}}$....	M. cu. ft.	4.4c.	3.7c.
133.	Selling price of coke-oven gas at gas holder (assumed).....	M. cu. ft.	12.0c.	12.0c.
134.	Net profit from sale of gas. Item 134—Item 133.....	M. cu. ft.	7.6c.	8.3c.
135.	Do. $\frac{\text{Item 116}\times\text{Item 134}}{1,000\times100}$ approx.....	24 hr.	$680.00	$1,000.00
136.	Do. Item 135×365 approx.....	Year	$250,000.00	$365,000.00

Cost of Blowing Blast Furnaces

137.	Total quantity of air blown (displacement of blowing tubs). Item 14×60 ×8760.....	Year	$36{,}800\times10^6$ cu. ft.		$47{,}300\times10^6$ cu. ft.
138.	Cost of installation of gas-blowing engine equipment complete (including one steam turbo blower) approx	.....	$650,000.00	100%	$650,000.00
139.	Subdivided into: Buildings.....	.....	$80,000.00	12%	$80,000.00
140.	Engine equipment....	.....	$530,000.00	82%	$530,000.00
141.	Gas-clean. plant.....	.....	$40,000.00	6%	$40,000.00
142.	Cost of installation on displ. basis (operating engines only). Item 138 : Item 14.	Cu. ft. displ.	$9.30		$7.20
143.	Do. on BHP basis (all installed engines at maximum continuous rating). $\frac{\text{Item 138}}{5\times\text{Item 29}}$	BHP inst.	$98.50		$75.00
144.	Fixed charges: 6 per cent. interest plus 7 per cent. amortization plus 2 per cent. insurance and taxes = 15 per cent. Item 138×0.15.....	Year	$97,500.00		$97,500.00
145.	Do. on displacement basis. $\frac{\text{Item 144}\times10^6}{\text{Item 137}}$	Mill. cu. ft. displ.	$2.65		$2.05

146.	Operating expenses incl. gas purification, all labor, repairs and maintenance, lubricants, water and miscellanous.....	Year	$40,000.00	$42,500.00
147.	Do. on displacement basis. Item 146 × 10^6 / Item 137	Mill. cu. ft. displ.	$1.09	$0.90
148.	Value of blast-furnace gas @ 10c. per mill. B.t.u. (see p. 73) $\frac{1{,}000 \times \text{Item } 5}{1{,}000{,}000} \times 10$.	M. cu. ft.	1.01c.	1.01c.
149.	Cost of coke-oven gas (see Item 132)	M. cu. ft.	4.40c.	3.70c.
150.	Value of 1:15 mixture of coke-oven gas and blast-furnace gas.....	M. cu. ft.	1.22c.	1.18c.
151.	Cost of fuel (blast-furnace gas only used) $\frac{\text{Item } 40 \times 365}{1{,}000} \times$ Item 148.....	Year	$43,000.00	$55,000.00
152.	Do. on displacement basis. Item 151 × 10^6 / Item 137	Mill. cu. ft. displ.	$1.16	$1.16
153.	Cost of fuel (gas mixture used). $\frac{\text{Item } 74 \times 365}{1{,}000} \times$ Item 150.....	Year	$42,800.00	$52,000.00
154.	Do. on displacement basis. Item 153 × 10^6 / Item 137	Mill. cu. ft. displ.	$1.16	$1.11
155.	Grand total cost of blowing blast furnaces (blast-furnace gas only used). Item 144 plus Item 146 plus Item 151..	Year	$180,500.00	$195,000.00
156.	Do. on displacement basis. Item 145 plus Item 147 plus Item 152.....	Mill. cu. ft. displ.	$4.90	$4.11
157.	Grand total cost of blowing blast furnaces (gas mixture used). Item 144 plus Item 146 plus Item 153	Year	$180,300.00	$192,500.00
158.	Do. on displacement basis. Item 145 plus Item 147 plus Item 154.....	Mill. cu. ft. displ.	$4.90	$4.06

159. *Recapitulation:* Cost of Blowing Blast Furnaces.

	Yearly	Per Mill. cu. ft. displ.	Yearly	Per Mill cu. ft. displ.
(A) Blast-furnace gas only used:				
Fixed charges.....	$97,500.00	$2.65	$97,500.00	$2.05
Operating expenses.....	40,000.00	1.09	42,500.00	0.90
Cost of fuel.....	43,000.00	1.16	55,000.00	1.16
Total.....	$180,500.00	$4.90	$195,000.00	$4.11
(B) Gas mixture used:				
Fixed charges.....	$97,500.00	$2.65	$97,500.00	$2.05
Operating expenses.....	40,000.00	1.09	42,500.00	0.90
Cost of fuel.....	42,800.00	1.16	52,500.00	1.11
Total.....	$180,300.00	$4.90	$192,500.00	$4.06

Cost of Producing Electric Power

(Gas-Engine Plant Operating jointly with existing Steam-Turbine Stations)

A. Blast-Furnace Gas only used for Power Purposes.

160.	Power plant capacity installed (see Item 55).....	Hr.	16,000 kw.		19,000 kw.
161.	Kw-hr. produced (see Item 66)	Yr.	100,000,000 kw-hr.		120,000,000 kw-hr.
162.	Cost of installation of gas-engine power plant complete.....	.....	$1,450,000.00	100%	$1,675,000.00
163.	Subdivided into: Buildings.....	.....	$160,000.00	11%	$185,000.00
164.	Engine equip	.....	1,180,000.00	82%	1,370,000.00
165.	Gas-clean. plant	.....	110,000.00	7%	120,000.00

166.	Cost of installation on kw. basis: Item 162 : Item 160	Kw.	$90 00	$88.00
167.	Fixed charges: 6 per cent. interest plus 7 per cent. amortization plus 2 per cent. insurance and taxes = 15 per cent. Item 162 × 0.15	Year	$218,000.00	$250,000.00
168.	Do. on kw-hr. basis. $\frac{\text{Item 167} \times 100}{\text{Item 161}}$	Kw-hr.	0.218c.	0.208c.
169.	Operating expenses incl. gas purification, all labor, repairs and maintenance, lubricants, water, and miscellaneous.	Year	$122,000.00	$135,000.00
170.	Do. on kw-hr. basis. $\frac{\text{Item 169} \times 100}{\text{Item 161}}$	Kw-hr.	0.122c.	0.112c.
171.	Cost of fuel. $\frac{\text{Item 68} \times \text{Item 148}}{1{,}000 \times 100}$	Year	$160,000.00	$192,000.00
172.	Do. on kw-hr. basis	Kw-hr.	0.160c.	0.160c.
173.	Grand total cost of electric power. Item 167 plus Item 169 plus Item 171	Year	$500,000.00	$577,000.00
174.	Do. on kw-hr. basis. Item 168 plus Item 170 plus Item 172	Kw-hr.	0.500c.	0.480c.

B. 1:15 *Gas Mixture of Coke-Oven and Blast-Furnace Gas used for Power Purposes*

175.	Power-plant capacity installed (see Item 92)	Hour	21,600 kw.		27,000 kw.	
176.	Kw-hr produced (see Item 103)	Year	140,000,000 kw-hr.		180,000,000 kw-hr.	
177.	Cost of installation of gas-engine power plant complete		$1,850,000.00	100%	$2,200,000.00	
178.	Subdivided into: Buildings		$200,000.00	11%	$240,000.00	
179.	Engine equip		1,520,000.00	82%	1,800,000.00	
180.	Gas-clean. plant		130,000.00	7%	160,000.00	
181.	Cost of installation on kw. basis: Item 177: Item 175	Kw	$86.00		$81.50	
182.	Fixed charges: 6 per cent. interest plus 7 per cent. amortization plus 2 per cent. insurance and taxes = 15 per ecnt. Item 177 × 0.15		$278,000.00		$330,000.00	
183.	Do. on kw-hr. basis. $\frac{\text{Item 182} \times 100}{\text{Item 176}}$	Kw-hr.	0.200c.		0.184c.	
184.	Operating expenses incl. gas purification, all labor, repairs and maintenance, lubricants, water and miscellaneous	Year	$147,000.00		$180,000.00	
185.	Do. on kw-hr. basis. $\frac{\text{Item 184} \times 100}{\text{Item 176}}$	Kw-hr.	0.105c.		0.100c.	
186.	Cost of fuel. $\frac{\text{Item 105} \times \text{Item 150}}{1{,}000 \times 100}$	Year	$220,000.00		$272,000.00	
187.	Do. on kw-hr. basis. $\frac{\text{Item 186} \times 100}{\text{Item 176}}$	Kw-hr.	0.157c.		0.150c.	
188.	Grand total cost of electric power. Item 182 plus Item 184 plus Item 186	Year	$645,000.00		$782,000.00	
189.	Do. on kw-hr. basis. Item 183 plus Item 185 plus Item 187	Year	0.462c.		0.434c.	

190. *Recapitulation:* Cost of Producing Electric Power.

(A) Blast-furnace gas only used:

Power-plant capacity	16,000 kw.		19,000 kw.	
Kw-hr. produced per year	100,000,000 kw-hr.		120,000,000 kw-hr.	
	Yearly	Per Kw-hr.	Yearly	Per Kw-hr.
Fixed charges	$218,000.00	0.218c.	$250,000.00	0.208c.
Operating expenses	122,000.00	0.122c.	135,000.00	0.112c.
Cost of fuel	160,000.00	0.160c.	192,000.00	0.160c.
Total	$500,000.00	0.500c.	$577,000.00	0.480c.

(B) Gas mixture used:

Power-plant capacity	21,600 kw.	27,000 kw.
Kw-hr. produced per year	140,000,000 kw-hr.	180,000,000 kw-hr.

		Yearly	Per Kw-hr.	Yearly	Per Kw-hr.
Fixed charges		$278,000.00	0.200c.	$330,000.00	0.184c.
Operating expenses		147,000.00	0.105c.	180,000.00	0.100c.
Cost of fuel		220,000.00	0.157c.	272,000.00	0.150c.
Total		$645,000.00	0.462c.	$782,000.00	0.434c.
191. Over-all cost of electric power adding to unit costs Items 174 and 189, respectively distribution cost of 0.4c. per kw-hr. associated with transformation, transmission, sub-stations, etc. Alternative A (blast-fce. gas only)	Kw-hr.		0.900c.		0.880c.
Alternative B (gas mixture)	Kw-hr.		0.862c.		0.834c.
192. Profit derived from sale of electric power at selling price of 1.2c. per kw-hr. Alternative A (blast-furnace gas only).	Kw-hr.		0.300c.		0.320c.
Alternative B (gas mixture)	Kw-hr.		0.338c.		0.366c.
193. Do. Alternative A (blast-furnace gas only). $\frac{\text{Item 66} \times \text{Item 192-A}}{100}$ approx.	Year	$300,000.00		$385,000.00	
Do. Alternative B (gas mixture) $\frac{\text{Item 103} \times \text{Item 192-B}}{100}$ approx.	Year	$475,000.00		$660,000.00	
194. *Recapitulation:* Profits derived from utilization of gases. (A) blast furnace gas only used for power purposes. Profit from sale of total coke-oven gas. $\text{Item 114} \times \frac{\text{Item 134} \times 365}{1{,}000}$	Year	$375,000.00		$525,000.00	
Profit from sale of electric power. Item 193-A	Year	300,000.00		385,000.00	
Grand total profit		$675,000.00		$910,000.00	
(B) 1:15 coke-oven and blast-furnace gas mixture used for all power purposes. Profit from sale of coke-oven gas. Item 136	Year	$250,000.00		$365,000.00	
Profit from sale of electric power. Item 193-B	Year	475,000.00		660,000.00	
Grand total profit	Year	$725,000.00		$1,025,000.00	
Saving of Alternative B over Alternative A	Year	$50,000.00		$115,000.00	

Conclusions

The foregoing calculations show that the installation of 450-ton blast furnaces offers distinct advantages over that of furnaces of 350 tons capacity.

It is found that the cost of blowing the larger blast furnaces is less per blast unit and, therefore, less per ton of pig iron produced than that of smaller furnaces and, furthermore, that the unit cost of producing electric power from the available surplus gas is considerably lower.

It is further shown that the unit cost of making coke-oven gas in a larger coking plant is a great deal less, and that the total revenues from

the utilization of blast-furnace and coke-oven gases are considerably greater with larger than with smaller blast furnaces.

The use of a mixture of blast-furnace and coke-oven gas for all power purposes is shown to yield substantial financial returns.

The cost of blowing the blast furnaces using a gas mixture for operating gas-blowing engines is exactly the same in the case of smaller blast furnaces, and, with larger furnaces, it is somewhat lower than when using straight blast-furnace gas for this purpose.

According to these calculations, the general use of such a gas mixture for power purposes results in a very handsome annual saving, especially when larger blast furnaces and a correspondingly larger coking plant are installed.

Discussion

Richard Lamb, New York, N. Y.—It is a fact that in Europe, especially in Germany, they have perfected the internal-combustion engines designed to use by-product coke-oven gas, and it is not uncommon to find as high as 5,000-h.p. units at work very successfully. In fact, I think the breakdowns and stops for repairs on these large engines in Germany are remarkably few. I have heard of a large engine running for a whole year without a shutdown for repairs.

In Germany they have had by-product coke-oven plants longer than we have had them here. In this country they have been installed mostly for use by steel plants, and possibly in three or four cases they have been built in connection with already established gas plants. These two kinds of by-product plants are radically different. One seeks to make coke and the other seeks to make gas as its principal product. If you try to establish a plant to furnish both gas and coke, it is a difficult matter to get suitable coal. A coal of about 28 to 30 per cent. volatile matter is about what is required; then you will get gas in quantity that will not make a soft coke, and the coke will be of a texture that can be sold to be used in place of anthracite coal. Where gas companies have established plants, the main object being to secure a large quantity of gas, they use a coal which makes soft coke. On the other hand, the steel companies are after a hard coke, and the gas production is but a small consideration and up to the present time as a rule is allowed to go to waste. The waste of gas at the steel manufacturing plants has been immense.

I was told that no one of the steel companies is using coke-oven gas for metallurgical purposes. However, I hear that the Krupp works in Germany is using the pure coke-oven gas in its open-hearth furnaces. I am told that, with a slight mixture of the blast-furnace gas with coke-oven gas, the Old Dominion Co. of Nova Scotia is using it very successfully in its open-hearth furnaces. The difficulty in using the coke-oven gas is due to the excessive heat it generates. Steel furnaces designed to

use blast-furnace gas are not built to withstand the extra heat of the coke-oven gas, but there is no reason why furnaces cannot be designed to withstand this extra heat. By-product coke-oven gas has about 500 B.t.u. per cubic foot, while the blast-furnace gas has about 90 or 91 B.t.u. per cubic foot.

I have noted that in attempts in this country to use this gas for engines the failures have been largely due to not providing sufficient cooling capacity. As far as I have been able to learn, the only engines in this country that have been built to use coke-oven gas are the two at Lebanon, Pa., and they did not meet the guarantee. The Nuremburg gas engine of Germany uses 8,700 B.t.u. per cubic foot of gas in producing 1 b.h.p. at full capacity load and about 9,300 B.t.u. for half load. In this country the open-hearth engines in Lebanon produced 1 b.h.p. with 11,000 B.t.u. per cubic foot and one-half horse-power with 14,000 B.t.u. per cubic foot. I do not recall now what their guarantee was, but the results were not acceptable, although the engine continued in operation. They have had trouble with the heating. Unless the United States Steel Corporation has put in a plant at Gary, which it intended to do, I do not think there are any other plants using strictly coke-oven gas in this country. I found, however, that the Westinghouse, the Snow, and the Allis-Chalmers companies are willing to make engines of large capacity and guarantee them at 10,000 B.t.u. per cubic foot per brake horse-power. You can buy engines on a guarantee of not to exceed 8,700 B.t.u. per cubic foot per brake horse power on full load, from the Nuremburg works of Germany. The Snow engine, I believe, could be made to show an efficiency of 8,700 B.t.u. per brake horse power on full load.

In establishing a by-product coke-oven plant to be operated independently of going steel or gas works, the question of what to do with the gas becomes a very serious one. If sold to gas companies you can get about 10c. per 1,000 cu. ft. for it, provided you will rectify it and bring it up to 18 to 21 candle power. The rectifying costs about 0.1c. per 1,000 cu. ft.; in other words, it brings the price for the gas down to about 9.9c. per 1,000 cu. ft., and a penalty is imposed if the quality of the gas goes slightly below the standard.

There are some very great advantages in selling the gas, independently of gas engines. You can transmit gas over long distances at very small cost, and you can sometimes get a market for it for fuel purposes, without rectifying it; and the requirement for the B.t.u. value will not be so great as if sold for illuminating purposes. Gas companies call for not less than 500 B.t.u., and demand a material penalty for going below that amount. It requires considerable operating care in a by-product coke-oven plant to keep the gas up to the standard, and great care in the purchase of the proper kind of coal is necessary in order to get satisfactory results. You should have coal with not less than 13,000 B.t.u. in order to

get 500 B.t.u. in the gas. In negotiating with a very large concern, that could take a large amount of gas, several million feet per day, I studied with their engineers the relative value of gas and coal for fuel purposes. With coal of about 13,000 B.t.u. costing $1.75 per ton, if I remember correctly, they figured that to make it advantageous to them, so as to justify their abandoning the coal and using the gas, the only contract that they could afford to make would be 7c. per 1,000 cu. ft. In computing the saving, the cost of handling ashes was not considered. I am convinced that at no distant date there will be many plants installed in this country to generate electricity with engines operated with coke-oven gas. Coal now known as coking coal—viz., that suitable for use in beehive ovens—is not necessary in making coke in by-product plants. The fine or pulverized coal, which is the cheapest, is needed in the latter ovens, and mixtures of different coals produce satisfactory results where neither by itself would be suitable. Ten per cent. more coke can be made out of a given quantity of coal with a by-product coke-oven than with the beehive oven. In the production of electricity there is a very great advantage in having a gas-storage tank. As soon as the service is cut off, the gas engine can be stopped and the tank stores the gas not used. In this particular it is like a water-power plant with a water-storage dam. In computing the total cost of the plant to generate electricity, I found that it compared very favorably with water power. At the same time that I was designing the coke plant, I was also designing a hydro-electric plant to generate 39,000 h.p. The cost, including amortization charges and interest on the plant and all costs, was 3.1c. per kilowatt-hour for water power and 3.2c. per kilowatt-hour for the gas, and the plant is for New York City.

I figured the power factor pretty high in the case of the gas plant, as we had to make only about 700 tons of coke per day, which gives us about 5,000 h.p. I figured on the basis of 70 per cent. power factor, because we were in a place where we could get considerable night service and some 20-hr. service, but that shows there is quite a conformity in the cost between hydro-electric power and coke-oven generated electricity. It also shows that with coal at $3.98 you can generate electricity for about the same cost as you can at a water-power plant, and that right in the very heart of large markets, where the transmission lines would be very short. For some time in Germany they have been making electricity at the mines and distributing it for 100 miles or more, and some of the companies engaged in this work have paid handsome dividends. In this country there are no plants making electricity from coke-oven gas, and there are billions of feet of gas going into the air, as dead waste, every day. In investigating the plant of the U. S. Steel Corporation, I mentioned that fact to my clients, and they tried to buy the wasted power. I understand that the company at once undertook to look into

the matter, and they are going to generate electricity and distribute it, if they have not already begun to do so. In Alabama, at Birmingham, there are millions of cubic feet of gas per day going to waste from the by-product coke-oven plants, but as it happens there is a surplus of power at that point. There are two big hydro-electric companies being built, the works of one being finished and the works of another well under way, and there is still another which may bring its power from Tennessee into the Birmingham district, and right there is all the coal they can use for centuries to come. Nevertheless it is a shame that this gas is being wasted. It is claimed that there are about 400,000 h.p. now generated with coal within a distance of 100 miles of Birmingham. The cheap power generated from gas and water now going to waste will ultimately supplant the present coal-generated power, and we can look for similar results from less-favored districts of the country.

A. E. Maccoun, Braddock, Pa.—I wish to ask Mr. Freyn whether he has any figures on the cost of operating a blast furnace as a gas producer. I think this data would be interesting, considering the differential between the value of coal and coke, and also I would ask whether it is a paying proposition to run a blast furnace as a gas producer.

Heinrich J. Freyn.—I am sorry I cannot give Mr. Maccoun any figures. Even if I had any, I could not give them, but I have none. I understand, however, that the cost of making blast-furnace gas compares very favorably with the cost of making producer gas; in other words, it is found at the gas plant that the use of producer gas made in one of the plants at the gas furnaces is distinctly a financially advantageous proposition, and this is very interesting from another angle. If you consider that the steel plant at Gary is dependent entirely on the use of gas power, and in view of the fact that a number of blast furnaces had to be blown out for well-known reasons, and that in addition the power plants had a little electric power to supply to industries in the vicinity, you can realize how important it is for the plant at Gary to have a solution of this problem. In spite of the fact that gas furnaces may be blown out in producing pig iron and steel, yet one of these furnaces using cheap coal can produce sufficient gas of such a quality that the power plant can be kept in operation just the same.

Richard Lamb.—An internal gas engine, as compared with the turbine steam engine, has a greater efficiency by 30 per cent.; in other words, an internal gas engine has 30 per cent. greater efficiency than the turbine.

J. E. Johnson, Jr., New York, N. Y.—Do I understand you run the furnace as a slagging gas producer, and charge it with bad coke, bituminous coal, and other cats and dogs?

Heinrich J. Freyn.—In South Chicago, where open-hearth slag was used, the flux necessarily resulted in a certain amount of iron, 60 or 70 tons per day, from a 500-ton blast-furnace. I understand that the pig iron so produced contains a great deal of manganese, and the credit for this particular iron has kept the cost of gas down.

J. E. Johnson, Jr.—The statement that one of these plants can compete with water power is, unfortunately, I am afraid, not correct. There are a good many chemical industries that require a great deal of power, and it seems as though it was the easiest thing in the world to hitch on to the blast furnace to get power and make a secondary profit from the waste gas, but one of these processes, which I happen to know about, figures that it must have power at $14 or better, in order to live. Very extensive figures have been published by Mr. Freyn, Mr. Bacon, Mr. Diehl and others, in connection with the development of gas power by blast furnaces, and these show that the actual cost of labor, maintenance, interest charges, and depreciation, etc., the absolute rock bottom charges, without any allowance for the value of the fuel at all, figure about 0.3c. per horse power-hour, and if you take 8,000 hr. to the year, that figures at about $24, which is prohibitive.

Heinrich J. Freyn.—It should be in kilowatt-hours.

J. E. Johnson, Jr.—That would reduce it below $20, but with a reasonable allowance for fuel it is still far beyond the price which many electro-chemical industries must have in order to live.

Richard Moldenke, Watchung, N. J.—In November last I saw in Germany the operation of the latest type of "slagging producer" made by Mr. Servais, and was much interested in noting the recovery of considerable quantities of ferro-silicon with the slag as it was tapped periodically. This material was the result of a reduction of the iron content in the ash of the coke breeze used, as well as from a considerable percentage of flue dust added to the fuel charges. The ferro-silicon ran about 12 per cent. in silicon, and should form a nice by-product to help bear the cost of gas production.

J. W. Richards, So. Bethlehem, Pa.—There were some papers read at a recent joint meeting of the American Institute of Electrical Engineers, the American Society of Mechanical Engineers, and the American Electrochemical Society, showing that power was being generated in the vicinity of New York City at 0.4c. per kilowatt-hour, in the large copper refineries, this being the total inclusive cost of the power.

Heinrich J. Freyn.—That is easy, without overhead charges, depreciation and interest.

J. W. Richards.—I understand that all that was included.

EDWARD T. CHILD.—I ask whether Mr. Freyn's contention would not operate more against the gas engine than against the turbine plant or steam plant? Is not the overhead greater in the case of the blast-furnace plant?

HEINRICH J. FREYN.—Yes, a little.

EDWARD T. CHILD.—Mr. Lamb's proposition was based on the engine itself without reference to the overhead. In the last plant I went through they had four large gas engines and five turbines. All the turbines were running, one-half of one engine was running, and the other three and one-half sets were out of service on that particular day.

J. W. RICHARDS.—I would ask Mr. Freyn whether this statement I find in his paper is quite correct: "Coke-oven gas used for power purposes, or as fuel in a steel plant, does not require any illuminating properties." I have had the impression that the illuminating property of gas was a very valuable property, increasing the radiation of heat from the gas, while it was being burned in the open-hearth furnace. For the same reason, if you extract the tar from producer gas, the gas is not nearly so efficient when used in the furnace, and I should suppose that the same would be true of coke-oven gas.

HEINRICH J. FREYN.—I must confess that I am not a metallurgical engineer. I know that the principal objection to the use of coke-oven gas in the open hearth was the lack of illuminating qualities, because the operator could not see the flame. If the tar that is recovered in the by-product plant were charged back into the coke-oven plant, it would be possible to recover the benzol, which is more valuable as a motor fuel to-day than gasoline, and that would be more advantageous than to burn it up in the open-hearth furnace.

Notes on Blast-Furnace Operation with a Turbo Blower

BY S. G. VALENTINE, OXFORD, N. J.

(New York Meeting, February, 1914)

BLAST-FURNACE blowing engines are broadly of two main types: either steam- or gas-driven reciprocating engines, or turbine-driven rotary engines. Some results of experience with a furnace blown by an engine of the latter type may be of sufficient general interest to relate. The first engine of this type was installed at Oxford Furnace, New Jersey, before the writer's connection with this plant, in March, 1910. This furnace uses chiefly New Jersey magnetites and had been in blast seven months on its old engines when the turbo blower was started. It was 80 ft. high by 17 ft. 6 in. bosh and 11 ft. hearth diameter. It is of interest to see from the records what happened when the turbo engine was put to work. But of course, as the old blowing equipment was rather antiquated, consisting of two I. P. Morris engines with air cylinders 72 by 72 in., in good running order, however, it is evident that the comparison between their work and that of the turbo blower cannot be interpreted as being between machines that were respectively of equal excellence as representatives of their types. Still the results were gratifying and interesting. The records show that, with the old engines blowing 13,630 cu. ft. of air per minute, measured by piston displacement, the furnace was working up about 480 tons of stock per 24 hr. and producing an average of 132 tons per day and a maximum of 166 tons.

The rotary blowing engine installed was a General Electric Co. machine, the driving end being a Curtis steam turbine and the air end a six-stage rotary compressor, all running at about 1,500 to 1,600 rev. per minute, and having a capacity of 25,000 cu. ft. of air per minute up to 25 lb. pressure, and equipped with an automatic constant-volume variable-speed governor. After running this blower, for a period equal to that covered above by the old engines, at an average rate of 15,000 cu. ft. of air per minute, the furnace worked up an average of about 592 tons of material and produced an average of 170 tons per day. Thus, on an apparent increase of 10 per cent. in the air volume there was an increase of about 23.5 per cent. in the materials worked up and of about 28.8 per cent. in the product. For the first calendar month's operation of the turbo blower, on an average of 15,040 cu. ft. of air per minute, there is

shown an average increase of 25 per cent. in the materials worked and of 33.3 per cent. in the product.

The furnace was blown out, relined, and put in blast again in June, 1912, and has been operating ever since. The only change in the furnace lines was an increase of 6 in. in hearth diameter, making it 11 ft. 6 in. Taking the operations of the last six months of 1913, to get the average of work done, the following data are of interest: The ores used are chiefly magnetites, averaging about 85 per cent. of these and 15 per cent. of other ores. Of the 85 per cent. magnetite, about 75 per cent. consisted of roasted magnetite carrying about 0.50 per cent. sulphur. The furnace runs chiefly on basic iron, and for this six months period had averaged 223 tons per calendar day, regardless of any time lost, with an average coke consumption of 2,147 lb. per ton of pig iron on an average consumption of 14,300 cu. ft. of air per minute, equivalent to an average of 41.7 cu. ft. per pound of coke used, the maximum being 43.5 and the minimum being 39.5.

Three main items of interest are developed in practice with our turbo blower. They are the low figures shown for quantity of air used per pound of coke burnt; the delivery of this air in a continuous stream instead of in pulsations; and the automatic regularity of the quantity of air delivered. Each of these items has its effect on the furnace operation, as further discussion will develop.

The item of the quantity of blast per pound of coke was the first result of the operation of the turbo blower to attract attention, because of what seems the very low figure shown. It required some time after blowing in this furnace for a Superintendent accustomed to the usual reciprocating engine quantities to adjust himself to the apparently low blast requirements in this case. The first impression was naturally that the thing was impossible, and that the rating of blast delivered was not correct. The makers of the engine, however, assured us that their calibration of the deliveries was guaranteed correct within 5 per cent. and they considered it actually correct within 3 per cent. It may be added that the blast lines and all connections from the engine to the tuyères are tight and entirely free from any noticeable leaks. Several inquiries as to the experience of other users of turbo blowers have elicited the information in one case that they were burning a pound of coke with 51 cu. ft. of air; in another case 53 cu. ft. were used, but with the added information that the blast lines were not tightly calked and there was a very noticeable leakage through them and through the air valves of the old blowing engines to which the cold-air main was connected.

In an endeavor to check this item we can calculate from certain data about what amount of blast is actually being blown into the furnace. The 2,147 lb. of coke per ton of pig iron, at 85 per cent. fixed carbon, contains 1,825 lb. carbon, and the 1,345 lb. limestone used per ton of

pig iron, at 11 per cent. carbon, contains 148 lb. carbon. The analysis of the furnace gases shows as follows: CO_2, 9.43; CO, 31.49; CH_4 + H, 0.90; N, 58.18; total, 100 per cent.

The CH_4 and the hydrogen were not separately determined, but as the quantity is small it does not materially affect the results.

	Pounds
Carbon from coke	1,825
Carbon from limestone	148
	1,973
Carbon to pig iron	90
Total carbon going to furnace gases	1,883

As there is 0.54 oz. carbon per cubic foot in either CO, CO_2, or CH_4, we find:

CO_2 0.0943 + CO 0.3149 + CH_4 0.0090 = 0.4182 × 0.54 = 0.225828 oz. carbon in gas per cu. ft.		
Carbon per cu. ft. gas, 0.225828 ÷ 16	=	0.014114 lb.
Gas per ton pig, 1883 ÷ 0.014114	=	133,413 cu. ft.
Nitrogen in gas, 58.17 per cent., would be per ton pig	=	77,606 cu. ft.
Air per ton pig, air being 79.3 per cent. N	=	97,860 cu. ft.
Air per lb. coke, 97,860 ÷ 2,147	=	45.5 cu. ft.

This compares with the engine record of 41.7 cu. ft. within about 8 per cent. and would indicate that the engine calibration is at least measurably correct. An element of slight inaccuracy may lie in the gas analysis used above, as the figures do not cover this period of six months and may not be an exact average of the whole period. But the analysis gives about our normal figures and the calculation seems confirmatory of the engine rating within a reasonable degree.

Assuming that the rating is approximately correct, we are impelled to a comparison with others in this respect. From general information, but without having received the actual data, we understand other users of turbo blowers are showing around 50 cu. ft. of air per pound of coke. On reciprocating engines the figures of one good modern plant, working on Lake Superior ores, were given me as 55 cu. ft Still another on the same make of blowing engines, working largely magnetite ores on foundry iron, shows 67 cu. ft. Another furnace with another make of reciprocating engines on magnetite mixture shows 57.6 cu. ft. Another furnace working on about two-thirds magnetites with still a third standard make of reciprocating engines used 62 cu. ft.

There are several factors that enter into any explanation of the variations shown as to the amount of air used in the above cases. In the first place, there is the item of the accuracy of the measurements. It seems that the quantities generally reported are lower on turbo blowers

than on reciprocating engines. It would be of interest to have the facts for other plants on both types of engine coupled with calculations of the air requirements based on the gas analysis, for comparison. Our own case seems to indicate reasonable accuracy as to the rating of delivery of air. In view of the fact that in most cases the ratings on reciprocating engines are considerably higher, how shall they be interpreted? We are inclined to charge part of the difference to a failure of reciprocating engines in many cases to deliver the quantities measured by piston displacement, and believe that in many cases furnaces are not actually getting the air they are supposed to get. It is a question whether cylinder capacities, clearances, character of air valve, and points of opening and cut off cover all the items in measuring the air deliveries. There is always the question as to the actual air admission on high-speed engines, and also of the delivery of the air against the varying tension in the blast main, on either high or low speed. Possibly the steady flow of air from a rotary blower as compared with the series of pulsations from a reciprocating engine may contain a factor that would furnish a partial explanation of the differences. I am willing to refer this line of discussion to those whose engineering ability better qualifies them to handle it.

A second feature to be remembered in interpreting these air quantities is the nature and physical condition of the ore to be reduced. Different ores will require different conditions in the consumption of coke in the furnace for their reduction and smelting. In certain cases the direct reduction by carbon in the upper part of the furnace will be greater than in other cases, and consequently the demand for burning carbon at the tuyères to CO will be less. On other ores again this condition will be reversed. J. E. Johnson, Jr., in a recent discussion[1] has called attention to this point with some very interesting data.

After eliminating the element of inaccuracy of measurements of the air used, the remaining explanation of any discrepancy between the actual air consumption and the amount theoretically needed to burn the carbon of the charge to CO at the tuyères must measure the loss of carbon in the upper part of the furnace, due to two causes—the waste of carbon by the action of CO_2, and the use of carbon for direct reduction of the ore. One would expect that as the former cause was the more active the requirement of coke per ton of pig would be relatively higher; and as the latter cause (direct reduction) was more active the requirement of coke per ton of pig would be relatively lower.

Taking our own case and assuming that our rating of 41.7 cu. ft. of air per pound of coke is approximately correct, let us compare the figure with what should be used if all our carbon were to be burned to CO at

[1] *Trans.*, xlvii, 351 (1913).

the tuyères. This amount is readily calculated. The carbon to be burnt per ton pig is 85 per cent. of 2,147 lb. coke, or 1,825 lb., less 90 lb. carbon for the pig iron, or 1,735 lb. As it requires 16/12 or 4/3 oxygen per pound of carbon burned to CO, and 100/23 as much air to furnish this oxygen, then air required, $1{,}735 \times 4/3 \times 100/23 = 10{,}058$ lb. Air at 60° weighs 0.0763 lb. per cubic foot, so that air required per ton pig is 131,821 cu. ft., and air per pound coke is 61.3 cu. ft.

Since the theoretical amount of air needed to burn 2,147 lb. of coke is 131,821 cu. ft. and the supposed actual quantity used is $2{,}147 \times 41.7$ or 89,529 cu. ft., this would represent a difference of 42,292 cu. ft., which divided by the theoretical requirement, 61.3, is equal to 690 lb. of coke, 32.1 per cent., or about one-third of the coke used per ton of pig, which in our case never reaches the hearth. Whatever explanation we find for this apparently large disappearance of carbon in the upper part of the furnace must accord with the by no means extravagant coke consumption given per ton of pig for a furnace working a refractory ore mixture. While doubtless there is a certain loss of carbon by action of CO_2 on the coke, it seems likely that there is an unusually large reduction of the ore by solid carbon above the tuyères. Otherwise we would expect to see higher coke requirement per ton of pig.

It looks as though this were an instance where by getting pretty far away from "Grüner's ideal working" we were effecting a coke economy by an unusual amount of reduction work being performed by fixed carbon. Prof. J. W. Richards has shown in his *Metallurgical Calculations*, p. 253, that "Whatever fixed carbon burns or oxidizes above the region of the tuyères, in a blast furnace, absorbs oxygen from the charges with three times the efficiency of carbon first burnt at the tuyères. . . . The ordinary furnace produces at the tuyères, in order to get heat enough to melt down the charges, *more* CO gas than is needed to abstract all the oxygen from the charge; under these conditions it is uneconomical to oxidize any carbon at all above the tuyères. The exceptional furnace, because of pure ores, small amount of slag, pure fuel, high temperature of blast, or dry blast, gets heat enough at the tuyères to melt down the charge *without* producing enough CO gas to reduce all the charges; under these conditions more or less reduction is effected by solid carbon, and with the greatest economy in quantity of carbon required in the furnace. These are the conditions under which, having passed the turning point, the greater economy of fuel is attained the farther away one can get from 'Grüner's ideal working.'" Under this explanation, in our case the credit for the result is probably due in large part to pure ores very uniformly well prepared physically and carrying a considerable quantity of granular fines exposing a large surface to reducing agents.

Returning for a moment to the case of furnaces blown by reciprocating engines, the following calculations are of interest. They are based in

the first two instances on data kindly furnished me, and in the third instance on data furnished in connection with A. N. Diehl's discussion of another topic.[2] The first two furnaces operate largely on magnetites and the third on Lake Superior ores:

	1	2	3
Lb. coke per ton pig	3,083	2,500	2,256
Lb. carbon in coke	2,635	2,150	1,897
Less carbon to pig	90	90	86
Lb. carbon to be burned per ton pig	2,545	2,060	1,811
Lb. air required if all burned to CO at tuyères, C × 4/3 × 100/23	14,753	11,942	10,498
Cu. ft. air per ton pig, lb. air ÷ 0.0763	193,355	156,513	137,588
Cu. ft. air per lb. coke, theoretical	62.7	62.6	60.9
Cu. ft. air per lb. coke, engine rating	67	57.6	60.8
Cu. ft. air per ton pig, engine rating	206,561	144,000	137,164
Excess engine rating in cu. ft	13,206	.....	.....
Deficit engine rating in cu. ft	.....	12,513	424

These figures show in the first case that, by engine measurements, 13,206 cu. ft. of air more than is needed to burn all the coke per ton to CO at the tuyères is being blown, an excess sufficient to burn 210 additional pounds of coke. This case leaves no allowance for any loss of carbon whatever in the upper part of the furnace.

In the second case the rated air delivery is within 12,513 cu. ft. of the quantity required if all the carbon were burned to CO at the tuyères, leaving only 199 lb. of coke per ton of pig to be lost in the upper part of the furnace.

In the third case the rated air delivery is practically exactly the theoretical requirement if all the carbon charged as coke were burnt to CO at the tuyères. This furnace seems to lose no carbon whatever in the upper part of the furnace.

One can only conclude either that these engine ratings of air blown are misleading, or that the furnaces in question get all their coke down to the tuyères without losing any of it in transit thither either by the action of CO_2 on it or by direct reduction of oxide by solid carbon

We next raise the question whether air delivered to a furnace in a continuous stream would have a tendency to cause different conditions within a furnace than those we are accustomed to consider produced by the air delivered in a series of pulsations. It is conceivable that these might be, less momentary lingering of the ascending gases, less time allowed them to play around the stock at any given part, and a more uninterrupted supply of gases at every point to act upon the descending stock. If there are any different conditions within the furnace traceable

[2] *Trans.*, xlvii, 396 (1913).

to this source they would naturally tend toward increased regularity of working.

Laying aside these aspects, we are free to say that there is a uniformity in our furnace operation that is not entirely due to uniformity of stock used, but for which we incline to give large credit to the uniform work of the turbo blower. Its automatic adjustment to pressure variations is prompt and effective, and so long as the boilers produce the necessary power we are not dependent on the personal attention of an engineer in adjusting engine speeds to meet the pressure changes on the furnace. It is an important aid to the uniform work of a furnace to have uniformity of blast volume, and to get it automatically is to get it most certainly. With uniform stock supplemented by uniform blast we are able to operate without changing our quantity of air for months at a time, and with no further changes of burden than the occasional substitution of one magnetite for another, and trifling occasional changes of limestone, but maintaining a fixed ratio of ores to coke. Whether its calibration be accurate or not, it certainly gives uniform quantities at any given rating, and whether we are actually using more or less cubic feet we are using the amount required with continuous and reliable regularity.

In conclusion, we believe that the turbo blower measures the volume of air used with more accuracy than the calculations based on piston displacement in other types of blowing engines; that as a corollary to this most furnaces run on reciprocating engines are using less air than they generally figure; also we are inclined to a preference for the steady stream of air furnished by a rotary blower, and are sure that the uniformity of volume automatically delivered, whether correctly or incorrectly rated, is an important feature in securing a uniform furnace operation.

Discussion

J. E. Johnson, Jr., New York, N. Y.—I did not have a chance to read Mr. Valentine's paper before the meeting and I would like to know how he reaches the figure of 41 cu. ft. of air per pound of coke. It is only two-thirds of the amount theoretically required and would indicate that one-third the total coke was lost by solution and never reached the hearth at all.

S. G. Valentine.—I would like to say, in reply to Mr. Johnson, that the figure of 41.7 is based on the calibration of the turbo blower. That calibration, I am advised, the General Electric Co. guarantees to be correct within 5 per cent., and they claim that it is really correct within a closer figure than that. Taking our gas analyses, as I have done in the paper, and where the figures are given, and figuring back from that I

came within 8 per cent. of that, my figures in the calculation being a little higher, running 45 cu. ft. of air per pound of coke charged. Of course, I am not able to certify to the correctness of that engine calibration, but understand it is based on very accurate and carefully made measurements.

J. W. Richards, So. Bethlehem, Pa.—I am not speaking as expert on the mechanical side of the turbo blower, but as I understand it, it is adjusted so as to take in a constant amount of outside air and deliver it to the furnace, so that its delivery is not a variable amount of air under variable pressure, but a constant amount of air under whatever pressure the furnace gives against it. The calibration, being done upon the inlet, is probably done very accurately.

Mr. Valentine's figures show that much less air is received by a furnace than it is ordinarily credited with. The old way of getting the amount of air going into the blast furnace, from the piston displacement, is a very rough and approximate way of getting at it, and we all know that the more accurate way is to take the analysis of the gases and work back from the balance sheet of the furnace, to find out how much air the furnace is receiving. This is more accurate than to take the piston displacement and make an allowance for efficiency. In fact, you can in that way compare the net delivery of a blower plant to the piston displacement of the engine. I have done that in many cases and it figures out, according to the modernness or the antiquity of the blowing engine, to an efficiency of delivery of anywhere from 85 down to 50 per cent.

The blast-furnace man who takes his piston displacement and tells you from that how many cubic feet of air he is blowing in per pound of coke, is giving you, probably, a misleading idea of how much air is entering his furnace.

The feature of blowing in a constant amount of air against the variable back pressure of the furnace is the most valuable thing about the turbo blower. The ordinary engine, without such regulation, gives the furnace the least amount of air when it needs the most; that is, when it is becoming stopped up and the back pressure is rising. The turbo blower is supposed to automatically deliver a constant amount of air under the variable pressures. We have therein an extremely valuable improvement in blast-furnace practice. I think it might be still further improved if the amount of air could be increased as the amount of moisture in the air increases (if the furnace has not a dry-blast plant) so as to compensate to some extent for the variable amount of moisture which is in the air.

I have calculated the volume of air received by Mr. Valentine's furnace, from the oxygen in the gases, and find it to correspond within

0.1 cu. ft., per pound of coke charged, with the value calculated by him on the basis of the nitrogen in the gases. This agreement checks up the accuracy of his data, and makes it in my opinion quite certain that his furnace was receiving just about 48 cu. ft. of air (at 60° F.) per pound of coke charged. If the regulator on the turbo blower registered less than this, I believe the difference due to inaccuracy in the calibration of the registering apparatus; if our reciprocating engine friends say that this is an impossibly small amount, the answer is that they have been too long accustomed to working from piston displacement, allowing too high coefficients of delivery, and so have become accustomed to charging their furnaces with more air than they were actually receiving. I consider this important point fully proved by Mr. Valentine's data.

J. E. Johnson, Jr.—It is perfectly true that the reciprocating blowing engines have efficiencies as low as 50 per cent. in some cases. I have made some tests along this line and I know that this is so. Nevertheless, I think that good engines such as are in common use now will show an efficiency of about 90 per cent. of the theoretical, including everything. It seems to me that the volumetric efficiency of Mr. Valentine's turbo blower must have been about 120 per cent., as I do not believe the results which Mr. Valentine reports can be obtained on a highly refractory burden when only two-thirds of the coke is being burned by the blast and the other third by oxygen from the ore. I think we must have more definite and positive information as to the nature and accuracy of the calibration of this apparatus before we should be asked to accept these figures.

Karl Nibecker, Youngstown, Ohio.—Mr. Valentine's paper as presented is of the most vital interest concerning the operation of a blast furnace when blown with a turbo blower. I regret exceedingly that up to the present time we do not have sufficient data available to enable me to make a direct comparison with the results obtained by Mr. Valentine. The machines which the company with which I am employed is at present operating have not up to the present time been tested sufficiently to enable me to give definite figures, but, from the data which we now have available, it appears as though the results which we are obtaining will not check in several respects with those as given in the paper.

Has Mr. Valentine ever calibrated the constant-volume governor as furnished by the builders of the blower? Has a Pitot tube, or some similar standard device for measuring air, ever been used in order to check the graduation of the regulating beam with the amount of air actually being delivered by the blower? It has seemed to me that it is quite important to carefully calibrate the measuring device as furnished with the blower with the machine actually operating under service conditions. It seems to me highly probable that the calibration furnished by the

builders of the machine under test conditions may not be accurate under working loads. The calibration may be quite different under actual operation when blowing a blast furnace.

It has been our experience that in many instances a sufficient correction factor for the slip and losses in a reciprocating engine is not assumed of a proper value, and this may account for the figure obtained by Mr. Valentine as to the amount of air delivered by the engine. It is also probable that the amount of air as figured for the turbo blower may be in error, even though a calculation of the air delivered to the furnace is made from the gas analysis and is found to check within a small per cent.

It is seen that the calculated amount of air required by the furnace is more than that delivered by the blower, if we accept the calibration of the regulator as correct.

We have noted conditions where the amount of air calculated from the gas analysis does not check with the air delivered by the engine after proper corrections have been applied to the engine, and I am therefore inclined to question the absolute value of checking the air delivered by an engine in this way, except as a most general indication.

I would like to ask Mr. Valentine whether he has made any comparison of the amounts of steam required per ton of metal produced by the two methods of blowing. It seems to me that this steam consumption is of the utmost importance in any furnace plant where an excess of steam can be made available for use in power generation. From our experience, it seems as though the turbine will deliver very nearly the same amount of air per pound of steam consumed as the reciprocating engine, providing that they are both working at the best vacuums for their respective design. It is probable that the turbine will consume slightly more steam than the engine per 1,000 cu. ft. of air delivered, but we do not have the data to state the exact ratio.

The type of governor with which our machine is equipped is the multiple Venturi tube of the Rateau design and seems to give a very constant flow. While the Venturi tube may be an accurate means of measuring air, we are inclined to believe that the setting of the regulator on this governor, of whatever type used, should be calibrated in place by some reliable means of measurement, such as the Pitot tube or throttling disk.

Leonard Waldo, New York, N. Y.—I have lately had occasion to get air into open hearths under varying pressures, and I have been very much troubled by the fact that, with a guaranteed pressure of 120 lb. on the mains, the pressure gauge of the main at the open hearth showed remarkable facility in dropping down to 50 and 52, considerably less than the pressure required for the service.

In the case of the turbo blower, I had hoped that we had gotten some-

thing in which the volume of air bore some relation to the pressure, for, after all, it is not the pressure which does the work, but it is the flame which takes place through the combustion of the fuel, and I think that in the minds of most people who are looking for air the turbo blower is the most promising thing, if the expense of the production of the air is not too great.

The open-hearth people have a strange faculty of translating these things into cost per pound of compressed air, and the people who put in these turbo-generator plants have a habit of placing the apparatus clear off at the end of the building, and driving them by belts, with the result that there is quite an increase in the charge. The air is very nice, but it costs like hot air generally does. Therefore, this paper is a move in the right direction, and I sincerely hope that the turbo blower is going to be a solution of the very great difficulties confronting us in the modern problems of combustion, in having adequate air in the right place at the right time and under the right pressure.

J. E. Johnson, Jr.—I do not wish to take up an undue amount of time in discussing this subject, but there is one point raised by Mr. Nibecker which is very important; that is, the probability that the turbo blower takes more steam than the reciprocating blowing engine. It is a more or less open secret in steam-engineering circles that from boiler pressure down to atmospheric pressure at least, the reciprocating engine is under no disadvantage as compared with the turbine in regard to economy, and a good many cases are on record where people have gone from turbines back to reciprociatng engines in power plants because they could obtain just as good economy and liked the operation better. The efficiency of the blowing engine, mechanically, is very high. Nordberg built some compressors which gave 95 per cent. mechanical efficiency, and I have in my possession some indicator cards from large blowing engines which show 92 per cent. A properly designed high-speed blowing engine has a high mechanical efficiency, and the thermodynamic or diagram efficiency of the air end is also high, whereas the turbo blower has an efficiency of only 70 per cent., which means, considering the two efficiencies together -that of the compressor and that of the engine—that the turbine must use less steam than the steam engine in the ratio of 70 to 90, and it is extremely doubtful if it can do this even with high vacuum. It seems to me that the opinion of the Youngstown Sheet & Tube Co., that the turbo blower requires more steam per unit of air delivered than good reciprocating blowing engines such as they have, is probably correct.

Karl Nibecker.—I would like to ask Mr. Valentine if he has noticed any difference in air pressure required at the furnace to produce a given amount of iron with the turbo blower as compared to the engine. I

have heard the statement made that the blower will produce the same amount of iron on less pressure than the engine, the theory being that the constant flow from the rotating machine will cause the same amount of air to be delivered through the furnace with less pressure. If this condition is true, it should be possible to blow a furnace with less horse power when a turbo blower is used than with a reciprocating engine. This would apparently indicate less steam per ton of iron. Whether this difference in air pressure and work done is enough to reduce the steam consumption on the turbine, I am not prepared to say.

If Mr. Valentine has any data concerning the steam, it will be of considerable interest.

J. W. Richards.—I would ask Mr. Johnson, assuming that the figures in Mr. Valentine's paper are accurate within 10 per cent., whether this does not show that this furnace was working with an excess of heat in the hearth and a deficit in the shaft, and thus explain the fact that carbon was burnt economically in the upper part of the furnace?

J. E. Johnson, Jr.—I should say it proves exactly the opposite.

George C. Stone, New York, N. Y. (communication to the Secretary*).—I fully agree with Mr. Valentine's conclusion that: "The turbo blower measures the volume of air used with more accuracy than the calculations based on piston displacement in other types of blowing engines; that as a corollary to this most furnaces run on reciprocating engines are using less air than they generally figure."

One reason for this he appears to have overlooked. In all of his calculations the temperature of the air is assumed to be 60° F. and its weight 0.0763 lb. per cubic foot. This is a fair assumption for the air in the engine room, but the cylinder, its heads, and the piston are at a much higher temperature, due to the heat of compression; consequently the air is heated on entering the cylinder, and the latter is filled with air at a temperature considerably above 60° and, therefore, contains a less weight of air than the calculation calls for.

The heat of compression is very considerable. Kent gives it as follows:

Gauge Pressure, Pounds	Temperature of the Air, Degrees F.
0	60
5	106
10	145
15	178
20	207
25	234

* Received Mar. 2, 1914.

The weight of air also varies considerably with its temperature, as follows:

Temperature, Degrees F	Weight of Air, Pounds per cu. ft.	Relative Weight
20	0.08274	108.3
40	0.07943	104.0
60	0.07638	100.0
80	0.07352	96.3
100	0.07090	92.8
120	0.06845	89.6
140	0.06617	86.6

The worst case Mr. Valentine gives shows a difference of only a little over 6 per cent. between the engine ratings and the theoretical. This would correspond to a temperature of the air in the cylinder of about 90° instead of 60°. This could readily be accounted for by the heating of the air by the piston, cylinder walls, and ports when working at very moderate pressures. This, I think, accounts for the fact stated in his corollary. As a further corollary it would follow that the higher the blast pressure the less the weight of air delivered.

A six-stage turbo blower should not be affected by the heat of compression to nearly as great an extent. As the compression in the first stage is very slight the heat imparted to the incoming air is also very slight. Aside from this the rating of the turbo blower is based on calibration, which would correct for errors due to this cause. This probably accounts for the more satisfactory work of the furnace with the turbo blower, as the volume and temperature of the air are constant regardless of the pressure, while with a reciprocating engine delivering a constant volume the weight decreases with increased pressure.

F. L. Grammer, Leesburg, Va. (communication to the Secretary*).—In Mr. Valentine's most interesting paper there is one item of carbon loss which he omits, as every one has done heretofore: viz., the amount blown out at the downcomer into the dust catcher. If he makes 25,000 lb. of dust a day and 10 per cent. is coke, there are 2,500 lb. of coke sent out of the furnace as dust, not as CO or CO_2 or in pig.

This is a small amount per ton, about 11 lb. per ton of metal, but would affect the final result a little.

It would affect it more in using unscreened coke from bells and using double bells, and where great volumes of wind are employed with high velocity of exit of gas.

S. G. Valentine (communication to the Secretary†).—In reference to the points raised in this discussion, I would say that a correction of the calculation of air from the gas analyses to a temperature of 60° F.,

* Received Mar. 28, 1914. † Received Apr. 15, 1914.

which was overlooked, gives as a result 48 cu. ft. of air instead of 45.5 as compared with the figures shown by the blower calibration of 41.7. This discrepancy of 6.3 cu. ft. may be due in part to the gas analyses thereby giving high results, or in part to any error of the calibration which might give low results. But even at 48 cu. ft., taking the highest figure, it appears that instead of the 131,821 cu. ft. of air required to burn all the carbon at the tuyères, only 103,056 cu. ft. of air were needed, indicating that 469 lb., or 21.8 per cent. of the 2,147 lb. of coke per ton pig, were consumed in the upper part of the furnace either by action of the ore or by the CO_2 of the gas. To my mind the economical coke consumption per ton of pig still finds its most likely explanation in the economical effect of direct reduction by carbon. The calculations of Dr. Richards referred to in the paper seemed to indicate that this is a fact.

The theoretical amount of air needed to burn a pound of carbon or of coke is easily figured. But in actual practice we know very well that we do not have to furnish this quantity for every pound of coke charged, for the simple reason that the coke does not all reach the hearth to be burned by the oxygen of the air. It is, therefore, self evident that any rating of air delivery which shows enough air blown to burn every pound of coke charged is an excessive rating. It seems, therefore, that in many cases less air is being actually used than the reciprocating-engine ratings would indicate. Mr. Nibecker states truly that sufficient corrections are not generally used in figuring on such air deliveries. I would add that some of the factors of these corrections are difficult, if not impossible, to determine accurately.

Relative to steam consumption by turbo blower, I cannot furnish any actual data. In our practice, operating with the condenser, the indications are that our steam consumption is not higher than it would be on a reciprocating engine of equal efficiency. Naturally, any blowing engine must have a sufficient steam supply to meet high furnace pressures. The turbo blower automatically speeds up at such times, thereby maintaining the supply of furnace gas under the boilers and therefore helping to maintain its steam requirements automatically. This is an advantage to the power equipment as well as to the furnace itself.

In regard to the pressure required for a given amount of iron with a turbo blower as compared to an engine, I do not think there is much difference to be noted. Pressure, being chiefly due to the resistance of the stock in the furnace, is probably less variable under a steady stream of air than it is under an intermittent or pulsating supply and might possibly average lower under the same furnace conditions. The fluctuations of pressure shown by a delicate gauge are less in the case of a turbo blower than they are with a reciprocating-engine air supply.

Turbo Blowers for Blast-Furnace Blowing

BY RICHARD H. RICE, WEST LYNN, MASS.

(Pittsburgh Meeting, October, 1914)

TURBO blowers for blast-furnace blowing have now been in use for some years, and a review of the experience gained and the present state of progress may be interesting.

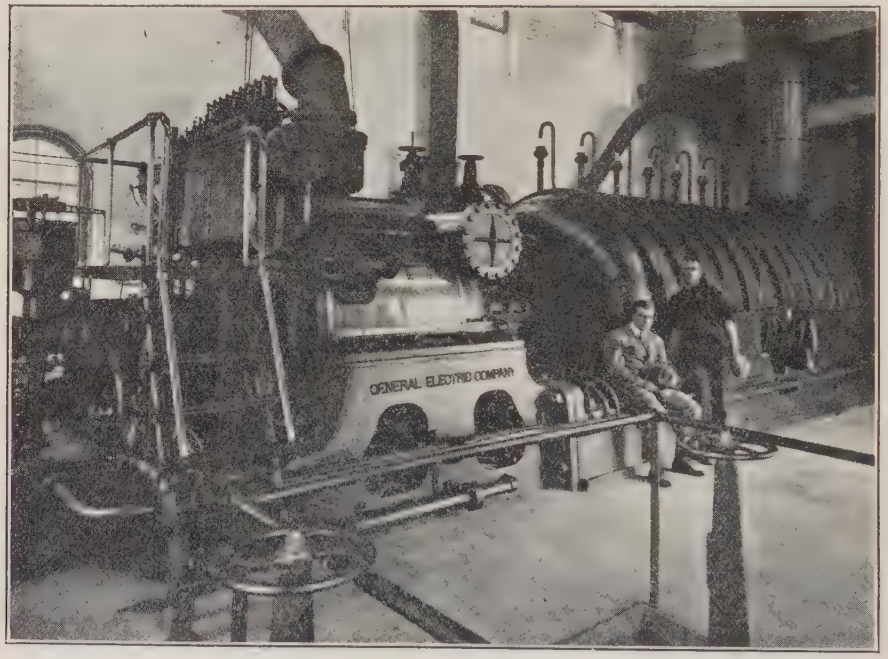

FIG. 1.—STEAM TURBINE DIRECT CONNECTED TO AIR COMPRESSOR.

The first blast-furnace blowing engine of this type to be installed in America was built by the General Electric Co. and is installed at the Oxford Furnace of the Empire Iron & Steel Co. This machine was put into operation March 8, 1910. A duplicate unit is situated at the furnace of the Northern Iron Co. at Port Henry, N. Y., which unit was put into operation June 30, 1910 (Fig. 1).

These two units consist of six-stage compressors driven by four-stage Curtis turbines, operating at variable speeds depending on furnace condi-

tions, the maximum speed being 1,950 rev. per minute. The rated volume is 22,500 cu. ft. of actual free air per minute.

Since then, besides these units, there have been built, installed, and put in operation by the General Electric Co., two 25,000 and four 45,000 cu. ft. machines; and by other builders, one 20,000, two 40,000, and one 50,000 cu. ft. machines.

Therefore, the total number of blast-furnace blowing engines of this type on which experience is available is 12.

Besides these, the General Electric Co. has built, installed, and has in operation 23 multi-stage turbo-driven air and gas compressors.

Method of Rating—Air Measurement

At the outset a radical difficulty presented itself. The rating of a reciprocating steam or gas blowing engine is usually based on displacement and this method is usually the basis of the determination (for practical purposes) of the amount of air being blown at any time. For the turbo blower, actual measurement of the air is the only criterion. If any other standard is used, an assumption is at once necessary as to the actual relation of the standard to the facts. The ratio between displacement and actual air delivered varies with many factors, as follows:

Clearance space in air cylinders (as pressure of air delivered increases more air is necessary to fill this space, decreasing air actually delivered).

Leakage past air pistons (increases with increase of air pressure).

Leakage past inlet and discharge valves (tends to increase with increase of air pressure).

Insufficient area of inlet valves (prevents cylinder from filling on suction stroke).

High temperature of air at beginning of compression stroke.

Insufficient area of discharge valves (prevents cylinder from completely discharging).

Waves in discharge pipe (same result as insufficient discharge valves).

Inertia of valves (has same general effect as insufficient valve area—in addition causes slip or back blow on return stroke).

Improper timing of mechanically driven valves.

From these causes, the actual air delivered may and often does amount to only 70 or 80 per cent. of the displacement.

On account of these discrepancies between rated and actual operating conditions, the first turbo blowers installed were found to be too large for the work. They were rated and proportioned to deliver normally 22,500 cu. ft. of free air per minute at 15 lb. normal or average pressure, with a maximum pressure capacity of 30 lb. As a matter of fact neither engine has ever blown over 18,000 cu. ft. per minute.

It was soon found that the steadiness of blast and perfect control which are the prime characteristics of this apparatus resulted in an increased output from the furnace and an improved quality; and these facts

have been noted in all similar installations where comparative data or experience can be had. The chief difficulty is to prevent overblowing of the furnace, since it is difficult for furnace men to think in terms of the new rating, and until they do, the furnace is always supplied with too much wind.

Difficulties Experienced

A number of difficulties have been experienced with these machines which are worth cataloguing.

The first difficulty met with was pulsation, or fluctuation of pressure, which occurs at volumes of say one-half normal rating and less (varying with the blast pressure). This pulsation consists of pressure fluctua-

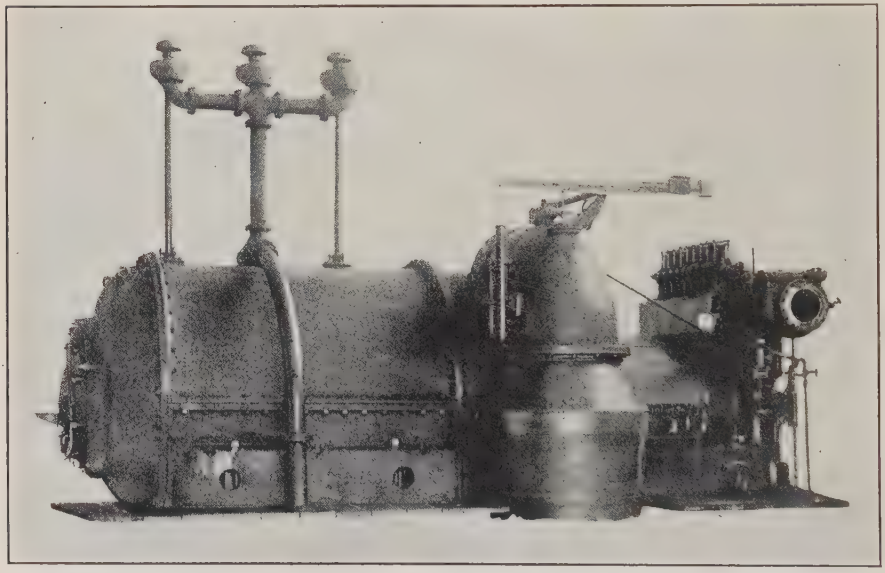

Fig. 2.—Centrifugal Compressor Direct Connected to Steam Turbine.

tions or waves of one or more pounds magnitude, and having a wave-length period of from one-half to one minute, depending on the volume of blast main and stoves which may be connected to it. This phenomenon has been entirely overcome in the machines which I have designed by the use of a blast gate (butterfly valve) in the suction pipe, between the air governor and the compressor. It is found that a very moderate throttling by this blast gate is sufficient to eliminate pulsation entirely (Fig. 2).

A difficulty met with in the first units built consisted in expansion of the bores in the hubs of the impeller wheels at maximum speeds, due to

insufficient allowance for forcing fits.[1] A mere statement of the difficulty indicates the proper remedy, which when applied entirely eliminated the difficulty.

Some of the turbines used have given trouble by throwing off buckets. This difficulty was finally traced to steam impulses corresponding to the free period of the buckets, and was eliminated by a change in the steam nozzles.

Southern furnaces, using leaner ores, which are besides of a sticky nature, and also using rather poorer cokes than their Northern prototypes, require more wind, and a higher pressure of blast. This severer service has brought out a weakness in impeller design. The entrance portion of some impellers, although of a stiff and rigid shape, has been found subject to cracks and breakage due to reed-like vibrations in the air currents, and these parts have been greatly strengthened and stiffened.

An accident to the turbine of a 50,000 cu. ft. compressor at Birmingham, due to the bursting of a turbine wheel, has been clearly proved to have been due to cracks produced in heat treatment, and not due in any way to defective design or inherent weakness. This machine will shortly be re-installed with all possibility of similar difficulty removed.

Relative Steadiness of Blast

I have frequently asserted that the blast from a turbo blower was appreciably steadier than that from a reciprocating machine, but this statement has often been denied by blast-furnace men, who have contended that the pulsations due to intermittent discharge from air tubs were smoothed out by the frictional resistances of blast main and stoves.

In order to set this question definitely at rest a delicate measuring instrument devoid of inertia was devised and constructed and records of pressure conditions at the tuyères of a blast-furnace were taken. Fig. 3 shows diagrammatically the apparatus which was used in making these tests.

The object of the apparatus is to show small variations on top of a heavy pressure (18 to 20 lb.) at the tuyère; it being well known that the ordinary instruments for measuring pressures of this magnitude have so much inertia, and are so far from being delicate, that all the variations, which it was the object of these tests to measure, are dampened out. The apparatus used is a differential recording gauge, consisting of a pressure diaphragm in a closed chamber, one side of which can be subjected to a constant pressure of desired amount, and the other subjected to the variable pressure which it is desired to measure, by means of a large pipe

[1] Sanford A. Moss: Increase of Bore of High Speed Wheels by Centrifugal Stresses, *Transactions of the American Society of Mechanical Engineers*, vol. xxxiv, p. 895 (1912).

capable of instantaneously transmitting the fluctuations. Motions of the diaphragm are transmitted to a delicate pen through a mechanism which is devoid of inertia, and the pen makes a record on a rapidly moving band of paper.

The diaphragm used was a thin copper disk about 14 in. in diameter, motions of which were taken from the center through a very light rod

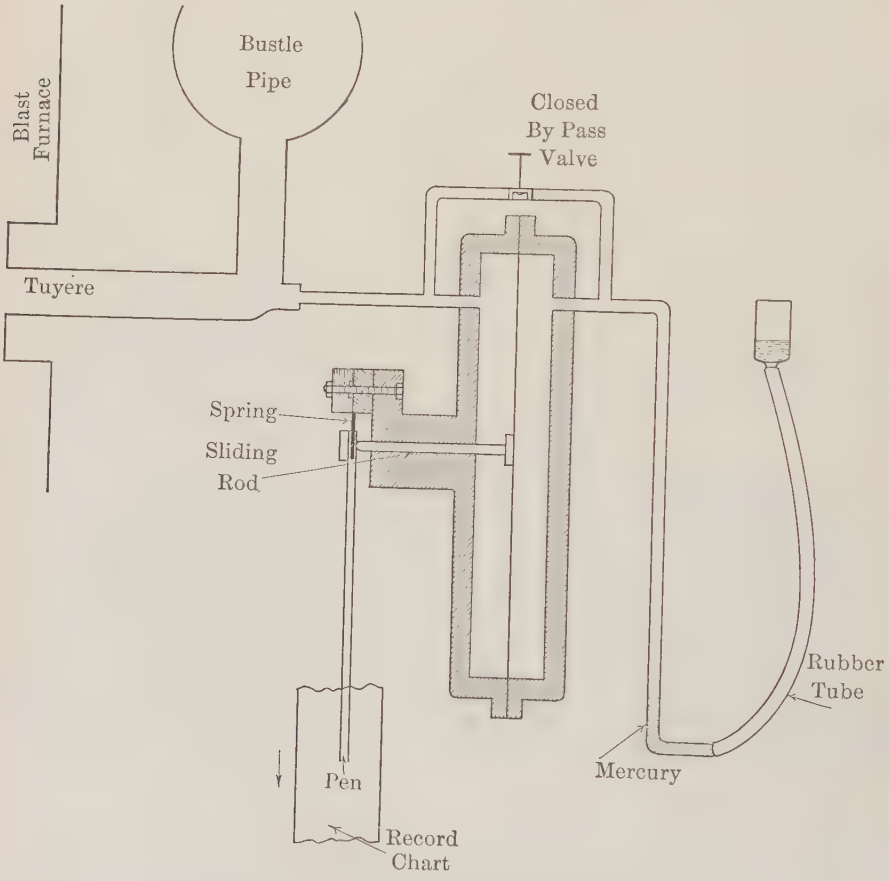

FIG. 3.—DIFFERENTIAL RECORDING PRESSURE GAUGE FOR MEASUREMENT OF PRESSURE VARIATIONS IN BLAST-FURNACE TUYÈRE.

sliding freely through a close-fitting bushing without appreciable leakage. This rod was pressed between the diaphragm on one end and the pen lever on the other, the fulcrum of the pen lever being a flat steel spring, so arranged as to press on the rod and through it upon the diaphragm, even with the greatest pressure fluctuation. The chamber on the back of the diaphragm was connected through a pipe of considerable size to a mercury **U,** consisting of a rubber tube, on the top of which was a reservoir for

mercury. The back and front sides of the diaphragm were connected by means of a by-pass pipe and valve, and the apparatus was operated as follows:

The by-pass valve was opened, connecting the two sides of diaphragm and equalizing the pressure. As soon as the pressure had been equalized, the by-pass valve was closed, and the pressure on the back of the dia-

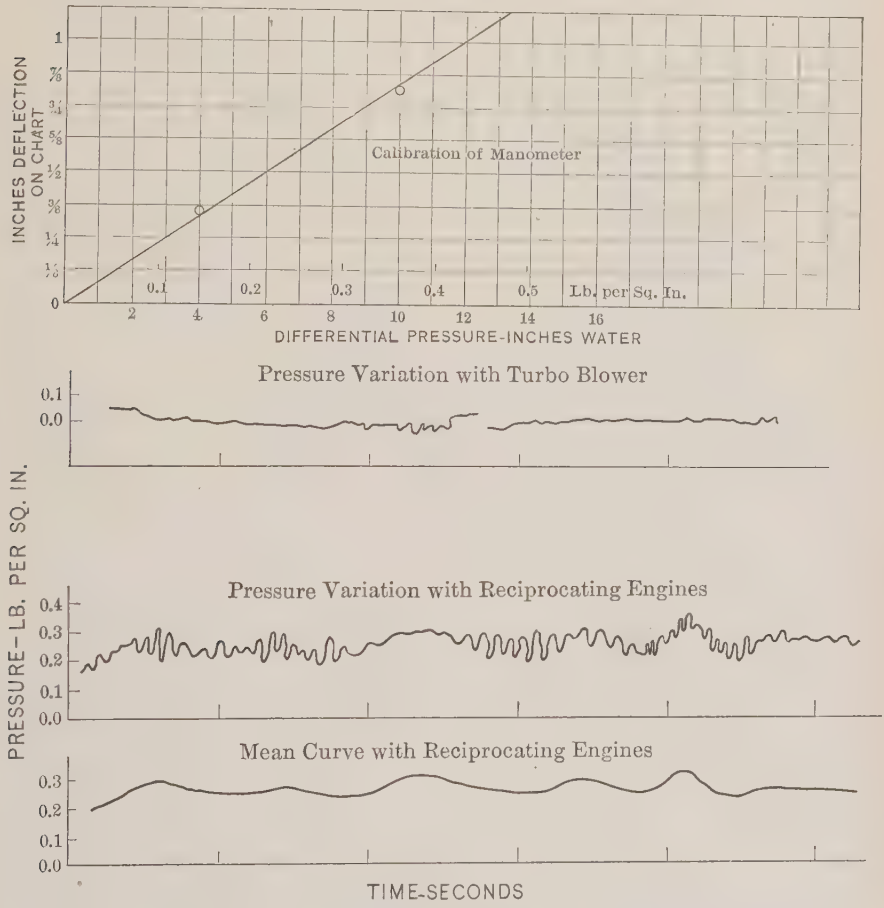

Fig. 4.—Curves of Pressure Variation in Blast-Furnace Tuyère.

phragm was adjusted by raising or lowering the mercury reservoir. As soon as this adjustment was complete the mercury reservoir was clamped in position. It was necessary that the by-pass valve, piping, and mercury tube be absolutely tight, and that the pipes containing the mercury be so large that the motion of the diaphragm did not appreciably change the capacity on the constant pressure side of the system.

The motion of the diaphragm was very small, the tension of the

pen spring on the diaphragm was considerable, and the inertia of the moving parts was as small as possible, resulting in a moving system, which was, to all intents and purposes, dead beat.

When operating the apparatus the chart was moved past the pen by hand, and the time interval was marked on the chart by an observer using a stop watch, marks being made at every 5 sec.

The records shown in Fig. 4 were made on a furnace working on Southern iron, and at the time it was blown with reciprocating engines there were four engines in use, two of which had steam cylinders 84 in. in diameter and air cylinders 84 by 60 in. stroke, while two had steam cylinders 44 in. in diameter. and air cylinders 84 by 60 in. stroke. All were of the vertical long crosshead type, equipped with Corliss air-inlet valves, and modified Reynolds type discharge valves, and were operated at an average speed of about 40 rev. per minute, the total displacement being about 60,000 cu. ft. of air per minute.

It will be noted that the curve obtained when operating in this manner is complex and apparently consists of a primary wave, and a secondary wave. Below the observed curve has been drawn one showing the primary wave, produced by tracing through the mean points of the secondary waves. These primary waves may be due to the discharge of the reciprocating cylinders, while the secondary waves may be caused by the fluttering of the discharge valves. In contrast to these is shown a curve which shows the pressure variations observed with the turbo blower.

Between the bustle pipe, where the observations were taken, and the engines, was a blast main about 1,000 ft. long and of large diameter, as well as the usual stoves. The use of this long and large blast main and four engines resulted in smaller fluctuations than would have been experienced with shorter blast main and with only one or two engines in use. It is hoped at a later period to make pressure observations of this arrangement also.

The method of calibrating the instrument was to produce deflection on the diaphragm by admission of air pressure to one side thereof so as to give a constant pressure difference, which was measured by a water U-tube, noting the deflection of the pen corresponding to these pressures, the chart given in Fig. 4 showing the result of this calibration.

Advantage of Steady Blast

It is believed that the superior steadiness of blast pressure from the turbo blower results in an increase of production, because the width of the melting zone is decreased, while the loss of fuel and ore due to the dancing of the particles on the blast and consequent attrition and blowing away as dust of such particles is reduced. That this is so is evidenced by the

reduction of the amount of dust caught by the dust catchers, which has been found in all such installations to be a very considerable percentage.

Since the width of the melting zone is reduced and all conditions of blast made more uniform, the blast can be used at a higher temperature.

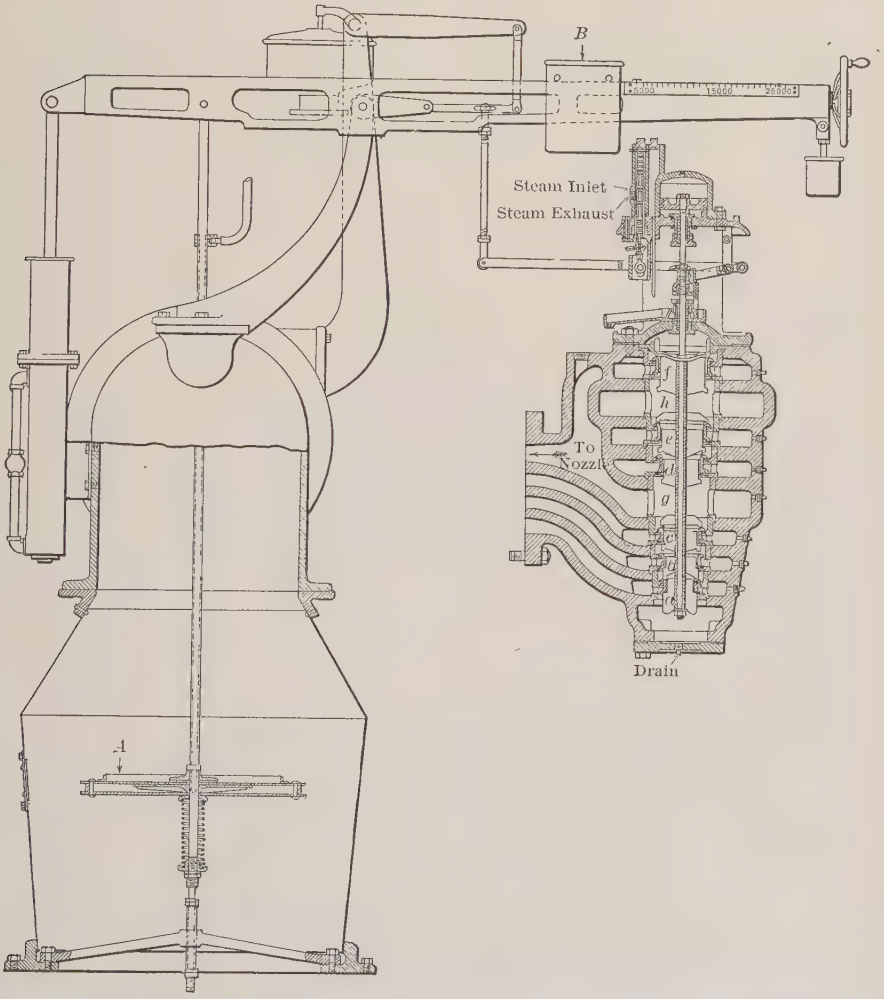

FIG. 5.—CONSTANT-VOLUME GOVERNOR FOR CENTRIFUGAL COMPRESSORS.

In practice, the use of turbo blowers, properly governed to measure the rate of blowing with accuracy, has the result of appreciably increasing the amount of iron produced, and this with a more uniform quality. This increase of production is accomplished without increase in the total coke charged and therefore the coke ratio is reduced.

There is evidence that this increase of production and decrease of coke ratio may amount to 10 per cent.

Governing Air Flow

The requirements for a governor to control properly the rate of blowing are ample power and accuracy.

The method used in apparatus built by the General Electric Co. is shown in Fig. 5. It consists of a disk, *A*, riding on the ingoing air

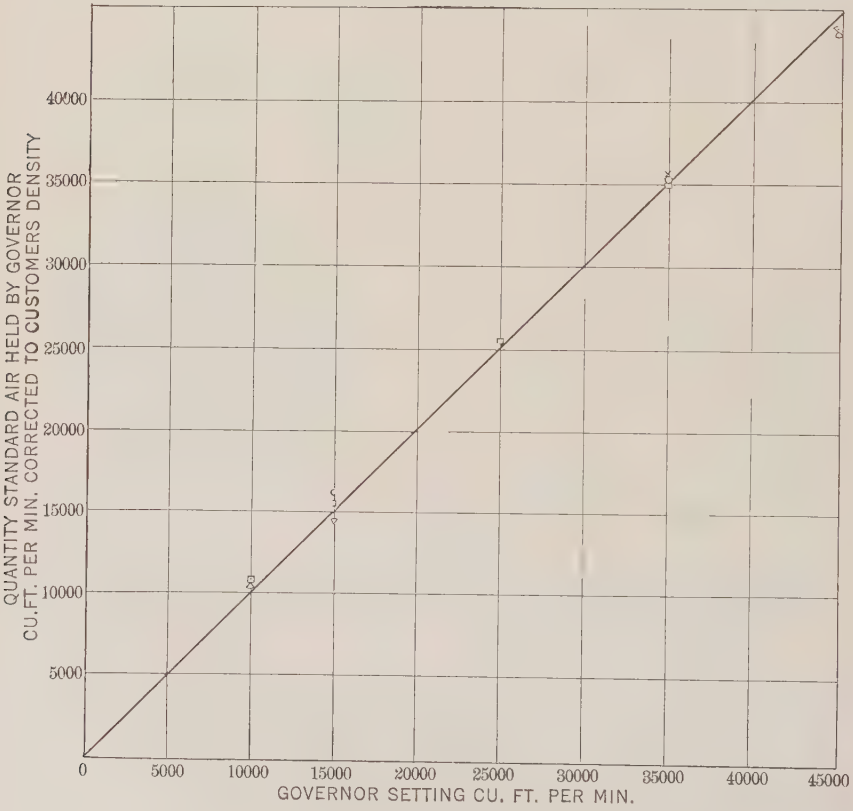

Fig. 6.—Calibration Curve of Constant Volume Governor.

in the center of a frustro-conical portion of the inlet pipe. Movements of the disk are transmitted direct to the valve gear, and ample power is secured by the use of very moderate dimensions. The angularity of the walls of the conical pipe is determined by the desired accuracy of regulation. No packing is necessary and the device is therefore practically frictionless and wear-proof.

Differing rates of flow are secured by moving a weight, *B*, along a scale

beam, which is graduated in cubic feet of free air per minute, and carefully calibrated at the factory before shipment. Fig. 6 gives a calibration curve of such a governor and shows the accuracy of the indications.

General Design of Turbo Blowers

Several methods have been used to balance end thrust in the air compressor. The method which I have used is to make all impellers double-entrance, without shroud rings, as shown in Fig. 7. While this method somewhat complicates the coring of the castings, it results in the maximum simplicity and durability of the rotor and eliminates any necessity for the use of a balancing piston, while the clearance packings

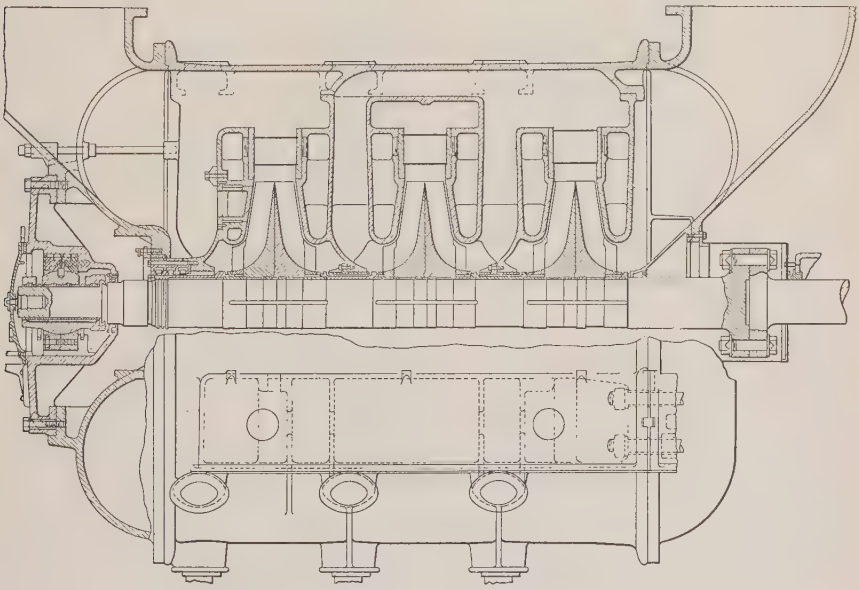

FIG. 7.—THREE-STAGE COMPRESSOR.

are on the shaft diameter and hence may have a large clearance with small leakage. Water jacketing is used in the diaphragms so as to cool the air while undergoing compression and in its passage from stage to stage.

Complete accessibility is secured by splitting casings and diaphragms on a horizontal plane. The turbine is similarly treated.

The locating bearing for the entire system of turbine and compressor is at the turbine end, and is so constructed that the axial position of the shaft with its turbine wheels and air impellers may be changed while the unit is in operation. (Fig. 8.)

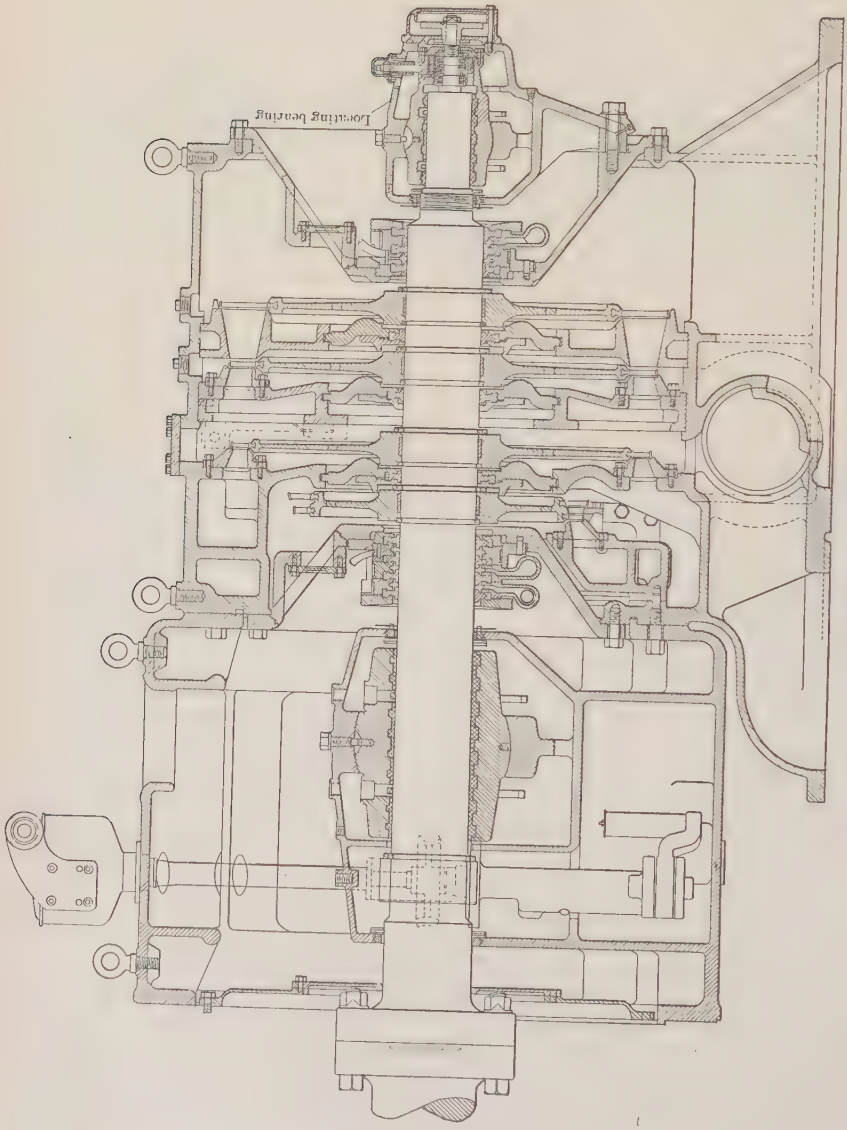

FIG. 8.—5,000-H.P. RIGID-FRAME STEAM TURBINE.

Comparative Efficiency

The relative thermal efficiency of this apparatus as contrasted with other forms of blowing engines is somewhat difficult to determine owing to lack of data. The reciprocating steam blowing engine has never been tested under circumstances which give any reliable basis for comparison, owing to the difficulty, amounting to impossibility, of accurately measuring the actual amount of air delivered per unit of time. The same thing holds true to a greater extent for the gas engine, since it is not only necessary to measure accurately the amount of air delivered for

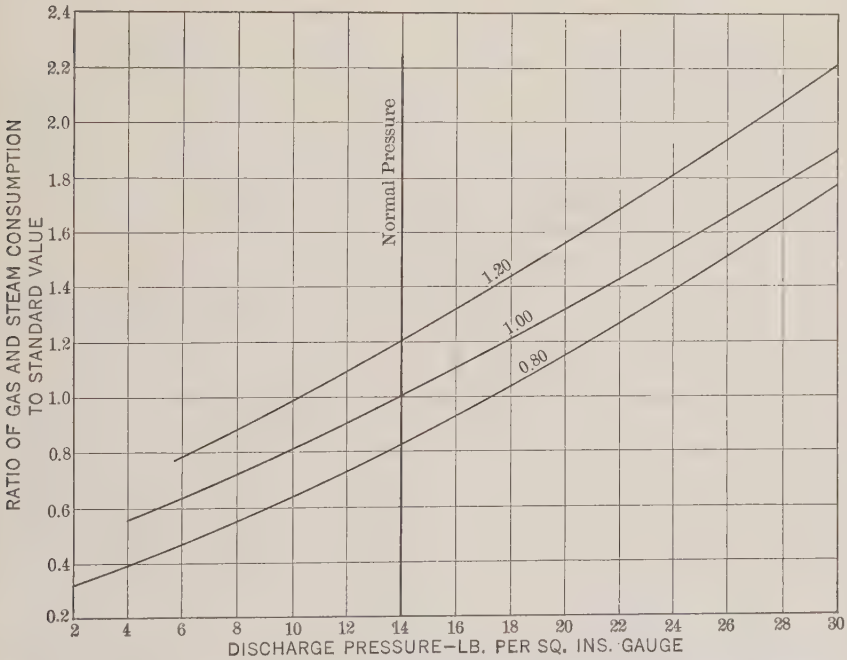

FIG. 9.—GAS AND STEAM CONSUMPTION OF TURBO COMPRESSOR WITH CONSTANT VOLUME GOVERNOR.

Figures on curves show fraction of rated quantity.

any length of time, but also the gas consumption. Both these measurements are subject to the same error. The source of this error is the fact that in all known methods of air metering, the indications vary as the square of the velocity of flow, and hence the mean of the readings is not the mean rate of flow. For this reason the readings of Venturi meters, Pitot tubes, etc., may give rates of flow as high as 40 per cent. in excess of the true value, depending on the amount and character of the fluctuations. We have, therefore, to fall back upon assumptions as to the ratio

of actual to displacement air, which are usually, due to this source of error, too favorable to the reciprocating engine.

From other data we can determine the comparative efficiency of turbine and steam or gas engine. These data are derived from tests of similar machines driving electric generators, on which much accurate information is available.

By observations of temperatures at inlet, between stages, and at discharge, together with accurate measurement of quantity and discharge pressure, and by determination (also quite accurate) of leakage and radiation losses, the efficiency of the compressor may be closely determined. This has been done, and curves showing the performance have been published.[2]

For the purposes of this discussion curves have been prepared (Fig. 9) showing the gas consumption of a turbo blower when blowing at various rates and against various blast pressures, from which the amount of gas needed under various conditions may be quickly determined. The gas consumption is of course that necessary to operate the plant.

Comparison of Steam-Engine, Steam-Turbine, and Gas-Engine Installations

In comparing the relative merits of these three types of blowing apparatus, the only proper method is to base the computation upon the best present practice for each.

The steam practice in blast-furnace installations is hopelessly behind the times, and in no way comparable with that of central power stations supplying electric light and street-railway service. In order to place the turbo blower on a fair basis for comparison with other types, it must be surrounded with the most efficient auxiliaries for steam generation, production of vacuum, etc., and blast-furnace engineers must make up their minds to abandon existing practice and follow in the footsteps of designers of electric power stations. To this end boilers must be installed capable of generating steam at 200 lb. pressure and with 200° superheat.

Where ordinary forms of gas burners are used under boilers it is not advisable to clean the gas, since the sensible heat, amounting to about 8 per cent. of the total heat in the gas, is available and utilizable. It will, however, be necessary to clean the boilers frequently. It is the practice abroad, and should be the practice here where efficiency is an object, to use pre-mixing burners, by the use of which the air can be accurately regulated and the best efficiency obtained. In this case it is advisable to clean the gas, otherwise the dust may slag on the tubes owing to the high furnace temperature.

[2] Richard H. Rice: Commercial Application of the Turbine Turbo-Compressor, *Transactions of the American Society of Mechanical Engineers*, vol. xxxiii, p. 381 (1911).

Furthermore, the practice of using a single jet condenser connected with a number of units, and receiving steam from as many as may be in operation, must be abandoned in favor of a condenser for each unit, preferably of the surface type, in order that the best possible vacuum may be realized for each unit unaffected by any misadjustment or improper operation of its neighbor. The use of a single turbo blowing engine per furnace has become standard practice on account of the small dimensions, lower cost, and higher efficiency per unit of capacity of large turbo blowers and the short time required to start a spare unit, to say nothing of the reduced cost of installation and operation of single units as contrasted with multiples. It is, therefore, desirable to lay out the turbo-blower house on the unit system, each blower having its own boilers, condenser, pumps, and other auxiliaries, with suitable spares. These various units will of course be cross connected by suitable pipes so that they may be used in all desirable combinations. In this way maximum flexibility is combined with the best efficiency.

Taking this method of installation as a basis for the turbo blower, we shall naturally use for the gas engine an installation of two units per furnace, since this plan appears to find favor with progressive gas-engine builders; and we shall also use similar arrangements of horizontal compound steam engines as the basis for the reciprocating steam-driven type.

For the boilers, we shall use the largest units possible to obtain, preferably two boilers per blowing unit, and of such type that the radiation losses from the brick work are a minimum, while the possibilities of decrease of efficiency due to lodgment of dust from the gas are as small as possible. The superheaters should be built in the same setting as the boilers for simplicity of manipulation.

Performance of Reciprocating Steam Engines

For plants as now being installed it is customary to use not over 150 lb. gauge pressure and not over 27 in. vacuum, and it must be admitted that this type of blowing unit is not well adapted for higher pressures, temperatures, or vacua in blast-furnace work. Therefore, we must assume a steam consumption for such engines of not less than 13 lb. per indicated horsepower under blast-furnace conditions, corresponding to 14.15 lb. per shaft horsepower.

As we shall see later, this assumption does not need to be exact, as the steam consumption would have to be reduced to a figure not attained in the best engines ever built for any purpose, with the highest degree of superheat ever used and with all conditions favorable for the steam engine (as they are not in blast-furnace work), to put this type on equal terms with the other as regards fuel consumption.

The data necessary for determining the gas consumption of this type of blowing engine are as follows:

Production of pig per day of 24 hr.	450 tons.
Coke per ton	2,240 lb.
Displacement per pound of coke (avg. value)	65 cu. ft. per min.
Theoretical horsepower required to compress 100 cu. ft. of air to 15 lb.	5 h.p.
Diagram factor for air tub[3]	0.94
Mechanical efficiency of air tub[4]	0.92

Displacement air required to produce 450 tons per day:

$$\frac{450 \times 2{,}240 \times 65}{24 \times 60} = 45{,}500 \text{ cu. ft. per minute.}$$

Shaft horsepower required to blow this air:

$$\frac{5 \times 455}{0.94 \times 0.92} = 2{,}630.$$

Boilers (coal fired) of large size have recently given 80 per cent. efficiency in electric power stations,[5] and even a small boiler without special attention has given recently 82 per cent. at the plant of the General Electric Co. at Lynn. For blast-furnace use, however, we must expect lower values, but 70 per cent. efficiency is readily attainable in practice with proper attention, or 75.5 per cent. with pre-mixing burners and clean gas. I shall assume the same efficiency, due to the same care, in case of turbo blowers and of reciprocating steam engines.

In the case of gas burned under boilers we have additional available heat due to the fact that we can utilize the sensible heat contained in the gas at high temperature. The amount of heat per cubic foot of gas per degree Fahrenheit is 0.0182 B.t.u., and allowing gas for boilers at 500° F. and for engines at 60° F., gives 8 B.t.u. sensible heat per cubic foot difference.

For this reason I take

	B.t.u.
Amount of heat in gas to gas engine	95
Amount of heat in gas to boilers	103

The heat per pound of steam is determined as follows:

	B.t.u.
Initial total heat	1,194.6
Final heat of liquid	83.1
	1,111.5

[3] Freyn and Bacon: Discussion, *Transactions of the American Society of Mechanical Engineers*, vol. xxxiii, p. 407 (1911).

[4] *Idem*, p. 408.

[5] J. W. Parker: Operation of Large Boilers, *Journal of the American Society of Mechanical Engineers*, p. 33 (February, 1914).

Hence we have the gas for boilers for reciprocating blowing engines:

$$\frac{2{,}630 \times 14.15 \times 1{,}111.5}{103 \times 0.70} = 574{,}000 \text{ cu. ft. per hour.}$$

Performance of Gas Blowing Engines

In the case of the gas-engine driven blowing engine we use the same data as above, to find the shaft horsepower required to blow the same quantity of displacement air. We also use the value of a horsepower-hour, which is

$$\frac{33{,}000 \times 60}{778} = 2{,}545 \text{ B.t.u.}$$

The thermal efficiency of the gas engine has been assumed in recent discussions[6] at 21 per cent., based on tests which were made with great care. Using this efficiency we shall consume

$$\frac{2{,}545}{0.21} = 12{,}120 \text{ B.t.u. per shaft horsepower in the gas engine.}$$

As above, we assume 95 B.t.u. per cubic foot of gas as the available heat for gas-engine purposes; hence we have for the gas required by an engine of this type to produce pig at the rate of 450 tons per day

$$\frac{2{,}630 \times 12{,}120}{95} = 335{,}000 \text{ cu. ft. of gas per hour.}$$

Performance of Turbo Blower

For the turbo blower the use of displacement air measurement no longer holds; and I have shown that the ratio between displacement and actual air is a variable, not a constant. The experience gained from the large number of units installed where air-governor measurements are available shows a variation in the air consumption per pound of coke from a minimum figure of 45 cu. ft. to a maximum of 55, depending upon local conditions, such as quality of coke, tightness of air valves and blast mains, etc. This experience teaches that 50 cu. ft. per pound is a safe and conservative figure for plants where all sources of loss are properly looked after and removed. Furthermore, I have shown the steadiness of the air current and consequent decrease of coke ratio due to the increased iron production resulting from this; however, I shall not in this discussion take any account of the effect of this, leaving it to future determination when more data are in my possession. We now have, for the turbo blower, the following data:

[6] H. J. Freyn: *Bulletin of the American Iron and Steel Institute*, p. 55 (1913).

Production of pig per day	450 tons
Coke per ton, as before	2,240 lb.
Actual air per pound of coke	50 cu. ft.

$$\text{Total air per minute } \frac{450 \times 2{,}240 \times 50}{24 \times 60} = 35{,}000 \text{ cu. ft.}$$

As is well known, the friction loss in blast mains, stoves, tuyères, and through valves is less for steady than for fluctuating flow. In the case of steam pipes it is common practice to make the pipes to steam turbines much smaller than in the case of similar pipes to steam engines. Also, the steady flow of air from a turbo blower maintains better conditions in the furnace and results in a freer flow through the charge.

For this reason, a lower blast pressure is needed with this type of blowing apparatus. This lower blast pressure of course decreases the power and gas consumption of the turbo blowers, but no account is taken of this item in the following computations:

We then have

Theoretical horsepower required to compress 100 cu. ft. of air to 15 lb	5
Making use of 200 lb. steam pressure, 200° superheat, and 28.5 in. vacuum (conditions actually obtaining in a number of electric power stations) we shall have steam consumption of turbine used as driver, pounds steam per shaft horsepower-hour	9.35

The shaft efficiency of the compressor has been assumed as 70 per cent., a result derived from many actual tests of such apparatus as safe and conservative.

We then have

$$\frac{5 \times 350}{0.70} = 2{,}500 \text{ shaft horsepower.}$$

The heat supplied to the steam by the boilers will be

	B.t.u.
Initial total heat	1,307.5
Final heat of liquid	60.1
Heat supplied by boilers per pound of steam	1,247.4

We can now determine the gas required for the boilers, which is

$$\frac{2{,}500 \times 9.35 \times 1{,}247.4}{103 \times 0.70} = 404{,}000 \text{ cu. ft. of gas per hour.}$$

We now have the following comparative results:

Gas Consumption in Cubic Feet per Hour

Reciprocating Steam Engine	Reciprocating Gas Engine	Turbo Blower
574,000	335,000	404,000

The table above shows that in order that the reciprocating steam blowing engine should equal the turbo blower in gas consumed it would be necessary to reduce its steam consumption to slightly over 8 lb. per indicated horsepower per hour, a result manifestly impossible to accomplish with any known practical apparatus.

I shall now discuss these figures in reference to the operation of a steel plant containing eight blast furnaces and ample use for electric power.

It is usually assumed that a 450-ton blast furnace produces 150,000 cu. ft. gas per ton of iron, or

$$\frac{150{,}000 \times 450 \times 8}{24} = 22{,}500{,}000 \text{ cu. ft. gas per hour}$$

in a plant of eight blast furnaces.

Of this 40 per cent. is consumed in stoves and leakage, leaving 60 per cent., or

13,500,000 cu. ft. gas per hour

available for blowing and the production of electric power.

I shall discuss two types of installations:

A—Blowing station with gas-engine blowing engines.
Electric station with gas-engine driven electric generators.
B—Blowing station with steam-driven turbo blowers.
Electric station with steam-driven turbo generators.

In the gas blowing station will be installed 20 gas-blowing engines. In the turbo-blower station there will be 10 turbo blowers.

In the gas-electric station, where the units will be of 2,000-kw. capacity, there will be 32 units.

In the turbo-electric station six 10,000-kw. units will suffice.

The comparative results on this basis are given in the following table:

Comparison of Turbine and Gas-Engine Driven Blowing and Electric Plants for Eight 450-ton Blast Furnaces

	A	B
1. Cubic feet of gas per hour required for blowing eight furnaces as per previous work for one furnace	2,680,000	3,232,000
2. Subtract (1) from 13,500,000 (the total gas available) to give cubic feet per hour of surplus gas	10,820,000	10,268,000
3. B.t.u. per cubic foot of gas (including sensible heat to boilers)	95	103
4. B.t.u. per hour in gas for electric station (surplus gas)	1,028,000,000	1,056,000,000
5. B.t.u. per kilowatt-hour (see next table)	18,820	25,200
6. Kilowatts producible in electric station from surplus gas	54,600	41,900
7. Total kilowatt capacity installed in electric station	64,000	60,000
8. Cost of electric station per kilowatt installed (using lowest figure[7] for gas electric station quoted in any publication but believed to be far too low for the ordinary purchaser)	$68.06	$50.00
9. Total cost of electric station	$4,355,840.00	$3,000,000.00
10. Shaft horsepower of blower per furnace (see above)	2,630	2,500
11. Shaft horsepower of blowing house as installed	26,300	25,000
12. Cost of blowing station (lowest published[8] for gas engine) per horsepower (rated) installed—actual figures for steam station	$74.50	$50.00
13. Total cost of blowing station	$1,959,400	$1,250,000
14. Total cost of plant, (9) plus (13)	$6,315,240	$4,250,000

In order that the turbo-electric station shall produce the same total kilowatt output as the gas-electric station it will be necessary to make, by coal burning

$$54{,}600 - 41{,}900 = 12{,}700 \text{ kw.}$$

To generate this will require

$$12{,}700 \times 25{,}200 = 320{,}000{,}000 \text{ B.t.u. per hour from coal.}$$

Tons of coal per hour at 24,000,000 B.t.u. per ton

$$\frac{320{,}000{,}000}{24{,}000{,}000} = 13.33.$$

At $1 per ton, as in the Pittsburgh district, this means a cost for coal of $13.33 per hour.

Cost of this make-up coal per year at $1 per ton:

$$365 \times 24 \times 13.33 = \$116{,}770.$$

[7] Arthur West: Discussion, *Bulletin of the American Iron and Steel Institute*, p. 65 (1913).

[8] *Idem*, p. 66.

I shall assume, as is done[9] at the Gary works of the U. S. Steel Corporation, 15 per cent. fixed charges for the gas-engine plants, and, as is the common custom[10] in large turbo-electric plants, 11.5 per cent. fixed charges for the turbo blower and electric plants.

Then we have

	A	B
Interest, depreciation, and taxes, per cent	15	11.5
Total fixed charges	$947,286	$488,750
Operating Costs of Blower House.		
Gas cleaning	35,200	.....
Plant operating and maintenance, labor and supplies	100,100	83,000
Water[11]	8,800	12,000
Total blower operating costs	$144,100	$95,000
Operating Costs of Electric Station.		
Gas cleaning	$105,600	.....
Plant operating and maintenance, labor and supplies	298,600	227,500
Water[11]	26,400	36,000
	$430,600	$263,500
Total operating costs and fixed charges per year	$1,521,986	$847,250

The difference between these operating costs

$$\$1,521,986 - \$847,250 = \$674,736.$$

The cost of make-up coal to bring up the output of the turbo-electric station to that of the gas-electric station at $1 per ton was found above to be $116,770.

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[9] H. J. Freyn, Gas Engine in Modern Blast Furnace and Steel Plants, *Bulletin of the American Iron and Steel Institute*, p. 47 (1913).

[10] This figure is customarily allowed by large public service corporations and is made up as follows:

	Per cent.
Depreciation of machines, 5 per cent., and buildings, 3 per cent., averaging	4½
Interest	5
Taxes and insurance	2
	11½

[11] The costs of water for the steam blower house and electric station "B" are taken high for extreme cases. Usually they would be greatly lower than the figures given.

It will be seen that the above difference in operating costs and fixed charges will not be wiped out until the cost of coal has risen to

$$\frac{\$674{,}736}{116{,}770} = \$5.78 \text{ per ton.}$$

The gas-plant costs have been based on an average kilowatt load substantially equal to that produced by the average gas. Actually this could not be obtained in the gas plant without some producers banked as reserve for periods of poor gas and peak loads. The cost, fixed charges, fuel costs, maintenance and operating charges of such a bank of producers have been wholly omitted. Such an arrangement would not be necessary in the case of a steam plant, as the boilers furnish sufficient heat-storage capacity and can be forced to any desired amount without additional plant. The cost of this reserve capacity must be added to the cost of plant, heretofore estimated, to put the two plants on an equality and will operate to increase the advantage of the turbine stations over the gas engine and increase the limiting cost of coal at which the two types become equal in operating costs.

The above figures take no account whatsoever of the improvement in quality and quantity of output from the furnace due to the use of the turbo blower, which will be all gain over the above figures.

The figures for the gas engine are based on the lowest obtainable. I believe them much too low for the average plant.

The figures for the turbo-blower and turbo-generator plants are taken on a very conservative basis.

Conclusions

From the foregoing the following conclusions are justified:

Turbo blowing engines give a steadier blast and a better measurement of blast than any other form of blowing apparatus for blast furnaces.

Steadier and better measured blast gives better conditions in the furnace, which leads to increased output and improved quality.

Turbo blowers operate with less attention and are more flexible in service than any other form of blowing apparatus for blast furnaces.

The gas consumption of the reciprocating steam blowing engine is so high as compared with that of a turbo blower, that it is not possible with any form of blowing engine in sight to come within reasonable range of the turbo blower in this respect.

Taking the best figures published by gas-engine manufacturers as representing the efficiency of the gas engine in practice and comparing them with the turbo-blower station in which modern steam practice is availed of to secure the efficiencies which are being obtained in turbo-electric stations, with reasonable allowances for the difference in fuels,

the gas consumption of the turbo blower is only 20 per cent. in excess of the gas blower.

Comparing blast-furnace and steel-works installations involving a plant containing eight blast furnaces with ample use for surplus gas and contrasting gas blowing and gas-electric stations with turbo blowing and turbo-electric stations (using the best figures given by any gas-engine advocates for the gas engine, and usual and conservative figures for the steam stations), the saving in operating costs and fixed charges by the use of the steam plant is so great as to make it necessary that coal should rise to $5.78 per ton in order that the cost of make-up coal to give the same outputs in the steam-electric and gas-electric stations could wipe out this saving; while with coal at present and usual prices, the saving by use of the steam turbine in such stations is very large.

The steam practice of our blast furnaces and steel works is hopelessly behind the times and needs entire revision, but with modern economical steam practice applied to such plants, the turbine-driven plant can be operated at lower cost with less stoppage and loss of product, and with a larger product for the same furnace of better quality, than by any other prime mover now in use for the purpose.

In practice, with a load factor of 75 per cent. in the electric station and 80 to 85 per cent. in the blowing house, the gas units will be operating below their maximum efficiency; and owing to their low mechanical efficiency and lessened thermal efficiency at light loads, will suffer considerably more under these conditions than the turbo blowers, which are designed for best efficiency at average operating conditions, or than the turbo-electric units, which have a remarkably good efficiency at light loads. Hence the above comparisons are unduly favorable to the gas-engine stations. Furthermore, neither gas nor turbo electric stations will generate sufficient power for the requirements of the steel mills to which they will be an adjunct; and the deficiency must be made up by additional units, the fuel for which must be coal. With gas-electric power units, gas producers must be installed; and since the requirements of a steel plant for power are very variable and the power station must be aways ready to supply the demand these producers will be banked for considerable periods; while the turbo-electric plant, by reason of the heat-storage capacity of boilers, will be always ready to meet the demand with little or no banking. Therefore, this part of the plant must inevitably be steam driven.

For smaller plants with fewer furnaces the advantage of the turbine-driven units becomes more and more considerable; and for plants with less than four furnaces, or for plants where use for the surplus gas does not exist, the turbo blower is without competition in adaptability to the blowing of blast furnaces by reason of its flexibility, quick adjustability to different furnace conditions, automatic control of pressure to maintain

a constant rate of flow into the furnace and steadiness of blast, resulting in improved furnace conditions.

Appendix

Computation of British Thermal Units per Kilowatt-Hour for Gas-Electric and Turbo-Electric Generating Stations

I. *Gas-Electric Station*

	Per Cent.
Maximum shaft efficiency	25
Average shaft efficiency during operation at good unit load factor	21
Average generator efficiency	93
Average over-all efficiency, $0.21 \times 0.93 =$	19.5
Allowing 8 per cent. of average load for auxiliaries, net average over-all efficiency $= 19.5 \div 1.08$	18.1

II. *Turbo-Electric Station*

	Maximum Efficiency	Average Efficiency at Good Unit Load Factor
Boiler	0.80	0.70
Generator	0.95	0.93
Turbine	0.75	0.70
Available energy (200 lb. per sq. in., 200° F., 28.5 in. vacuum		398.3 B.t.u.
Heat added to steam in boiler		1,247.4 B.t.u.

$$\text{Over-all efficiency} = \frac{398.3 \times 0.70 \times 0.70 \times 0.93}{1{,}247.4} = 14.6 \text{ per cent.}$$

Allowing 8 per cent. of average load for auxiliaries,

net average over-all efficiency $= 14.6 \div 1.08 = 13.5$ per cent.

Hence the following figures, for electric stations with good load factors such as would occur in such plants, allowing for all auxiliaries, are per kilowatt-hour:

Gas-engine station ..... $3{,}412 \div 0.181 = 18{,}820$ B.t.u.
Steam-turbine station ..... $3{,}412 \div 0.135 = 25{,}000$ B.t.u.

Discussion

S. G. Valentine, Oxford, N. J.—In a paper read at the New York meeting of the Institute last February I presented some notes taken from our experience with the turbo blower operating on a blast furnace. In

connection with Mr. Rice's very interesting paper there are a few items that might be of further practical interest.

The question of the reliability of this type of blower will naturally be presented. In this connection we would say that our turbo blower has been in practically continuous use during the present blast, which was begun June 7, 1912, with only three interruptions for repairs of any consequence. Soon after blowing in there was developed a tendency of the blower to surge or pulsate. This occurred on operating at a point considerably below the normal capacity of the machine, and was checked with comparatively little trouble by throttling the intake slightly and taking up a small lost motion on the governor connections, a legacy of the previous blast that had not been attended to. All this required only a few hours' time. In March, 1914, the furnace being banked on account of a blizzard that cut off our freight deliveries for five days, the opportunity was used to replace the carbon packing on the steam-turbine shaft and to repack the steam inlet valves. In June, 1914, the blower was off 8 hr. to replace an oil ring and scrape a bearing. With these exceptions we have made continuous use of the machine and find we can depend upon it with entire safety as a piece of machinery.

It is reliable also in regard to its air deliveries. Being the first one built this blower was designed to throw a larger volume of air than we are able to use and is therefore run at below its normal rating, and at our running speeds the actual quantity of air furnished is probably a trifle more than the calibration would indicate. This, in our case, might amount to 1 or 2 per cent. However, the regularity of the amount delivered is evidenced by the general regularity of the furnace operation, and we find we can rely on prompt and automatic adjustments to pressure variations, maintaining thereby a constant volume. It is rather difficult to show either graphically or in figures the element of furnace regularity; and, of course, in addition to variable blast volume there are a number of other things that might affect the regular working of the furnace. Eliminating these items, however, there is evidence in our daily experience of very regular driving and freedom from any tendency to slip or hang, as well as regularity of quality in our product, for which we are inclined to give very much credit to regularity of blast used. When running on basic iron it is not unusual for the furnace to make 90 to 93 per cent. of standard basic on a magnetite mixture, 75 per cent. of which carries sulphur to the extent of 0.50 to 0.70 per cent., which is quite satisfactory.

While not able to present any formal statements as to cost of operating and maintenance as compared with the other reciprocating steam or gas engines the figures for our own case may be of interest. For all repairs, including packings of all sorts, all pipe fittings, etc., used directly and indirectly during the last 28 months, there has been expended

$227.15. For oil, waste, and all similar supplies the amount for the same period has been $159.22, or for both above items slightly more than two mills per ton of pig iron produced.

Mr. Rice's paper includes a very interesting description of the tests made to determine the difference in the vibration of the air deliveries from rotary and reciprocating engines. The results obtained by him seem to bear out what our experience would have led us to expect in this particular. I am personally convinced that a continuous stream of air is better than a pulsating stream of air as far as the smoothness and regularity of furnace operating is concerned. It would be perhaps difficult to specify any particulars observable in regard to this item. But certainly whatever difference may be shown to exist as between these two types of air-current deliveries to the tuyères would be in favor of the most regular delivery obtainable, which seems to be secured in the highest degree from a rotary engine.

W. McA. Johnson, New York, N. Y.—The question of turbo blowers vs. piston blowers can be considered, in my judgment, entirely in the light of engineering economics. This gives the verdict to the turbo blower entirely on the capital cost, along the following lines of reasoning:

We can admit that for pressures of 25 lb. per square inch the piston blower has a somewhat better volume efficiency, say 85 per cent. as against 75 per cent.; better mechanical efficiency; and makes less flue dust in smelting than the turbo blower. But the capital cost of the turbo blower must sooner or later prove to be one-third to one-half that of the piston blower, remembering that modern high-speed turbo generators sell in large sizes for as low as $10 per kv.a., or even lower, as against a far larger sum per unit for the slower-speed reciprocating equivalent.

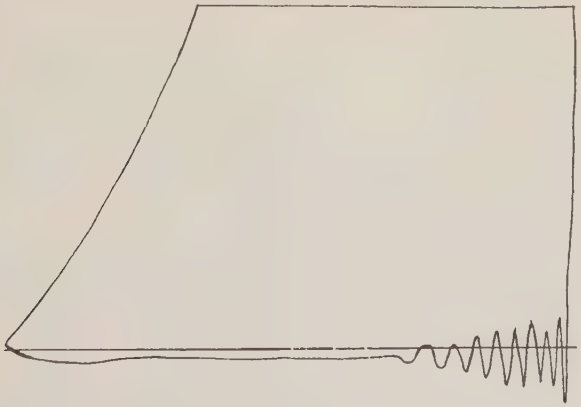
It is reasonable then to suppose that eventually turbo blowers will sell for "so much per pound." Then a great deal of $\frac{1}{2}$ mv^2 is raced through a few tons of iron, steel, and brass in the case of the turbo blower. Now when new construction of blast furnaces does come, as come it will, the turbo blower will appeal to bankers and to such engineers as partake of their views, for reason of its low capital cost as compared with that of the reciprocating blower.

Air is cheap and we can waste it. So is power and we can waste it. But capital is scarce and we must conserve it. The advance in the state of art is now so rapid that "commercial obsolesence" demands high charges for depreciation in metallurgical industry. It is good finance that places this at a minimum of 15 per cent. per annum. To my mind, this consideration is all important in making a comparison of the future of the turbo blower. The commercial history of the centrifugal vs. the reciprocating pump has shown the same to be true in the past.

L. Iversen, Pittsburgh, Pa.—I would like to give you a few figures showing a comparison between the turbo blower and the modern re-

ciprocating steam blowing engine of the latest type with blowing cylinders equipped with automatic plate valves, and I will place the steam-engine driven reciprocating blower on exactly the same basis as the turbo blower described by Mr. Rice, both in respect to the amount of air delivered and the steam and vacuum conditions.

I will not discuss the question of how much air is required to burn a pound of coke in the blast furnace, but I believe I am safe in saying that it will require the same amount of air for equal conditions in the furnace, whether same is supplied through a reciprocating or a rotary machine. I will base my figures on 65 cu. ft. of air per pound of coke, or 45,500 cu.



Crank end of cylinder, low-pressure air side. Cylinder 84 in. diameter; 60 in. stroke; 57 revolutions; boiler pressure 130 lb.; vacuum 26 in.; blast 20 lb.

FIG. 10.—INDICATOR CARD FROM RECIPROCATING BLOWING ENGINE.

ft. per minute for the same size furnace as discussed by Mr. Rice, which will give us a theoretical air horsepower of:

$$\frac{45{,}500 \times 5}{100} = 2{,}275.$$

The modern blowing engine *cylinder* will give a compression efficiency of 95 per cent. and a mechanical efficiency of 97 per cent., which would give a horsepower at the piston rod of:

$$\frac{2{,}275}{0.95 \times 0.97} = 2{,}470 \text{ hp. for the engine,}$$

against a shaft horsepower of:

$$\frac{2{,}275}{0.70} = 3{,}250 \text{ hp. for the turbo blower.}$$

An indicator card taken from a blowing cylinder of this kind is shown herewith, Fig. 10. This card was taken with a five-scale spring and in the

most careful manner. It will be noted that the suction line rises above the atmospheric line at the end of the stroke. This card shows a volumetric efficiency of greater than 100 per cent. and the maximum suction below the atmosphere is 0.18. After deductions have been made for rise in temperature and valve resistance, we get from measurements a compression efficiency of 95 per cent.

If we substitute 3,250 shaft horsepower for the turbo blower instead of 2,500 and accept Mr. Rice's figures of 404,000 cu. ft. of gas per hour for 2,500 hp., we get a gas consumption for the turbo blower of:

$$\frac{404{,}000 \times 3{,}250}{2{,}500} = 525{,}000 \text{ cu. ft. per hour.}$$

In order to utilize the available energy assumed for operating the turbo blower to advantage in the steam engine, namely, 200 lb. pressure, 200° superheat, and 28½ in. vacuum, we will install cross-compound engines and operate same non-condensing with a low-pressure turbo generator which will take the exhaust steam from the engines and readily get the benefit of the 28½ in. of vacuum.

The steam consumption of this type of engine per indicated horsepower per hour will be 17 lb., the mechanical efficiency 88 per cent., and the steam consumption per hour:

$$\frac{2{,}470 \times 17}{0.88} = 48{,}000 \text{ lb.}$$

The steam consumption of the low-pressure turbine will be 27½ lb. per kilowatt and the kilowatt output:

$$\frac{48{,}000}{27.5} = 1{,}750.$$

Assuming 25,200 B.t.u. per kilowatt (which I believe is far too low) and 103 B.t.u. per cubic foot of gas, we get:

$\frac{1{,}750 \times 25{,}200}{103}$ = 430,000 cu. ft. of gas, which amount is to be credited to the total amount consumed by the steam-engine driven reciprocating blower. This is using:

$$\frac{48{,}000 \times 1{,}247.4}{103 \times 0.70} = 830{,}000 \text{ cu. ft. per hour.}$$

Deducting 430,000 which we are getting back in the kilowatt output, we get a gas consumption chargeable to the steam-engine driven reciprocating blower of 400,000 cu. ft. per hour, which is equivalent to producing an indicated horsepower in the engine on less than 8½ lb. of steam per hour considering the combined installation.

From the above it will be seen that the turbo blower uses 125,000 cu. ft. of gas, or 32 per cent., more than that required to operate the

reciprocating blower, and since the cost of installation of the steam-engine driven reciprocating blower and the turbo blower will be approximately the same, it is evident that the turbo blower cannot compete with the former for blast-furnace work, no matter how well the structural difficulties have been overcome.

You will note the tremendous difference in the results obtained by me and those given by Mr. Rice, and since the point at issue is of great importance, I would suggest that a commission of disinterested parties be appointed to observe the results, witness and make tests in order to determine whose assertions are correct. As for myself, I cannot accept Mr. Rice's methods of measuring the air to the turbine and by means of the theoretical horsepower required to compress this air and the efficiency of the machine, to get the steam consumption. I would measure the steam going to the turbine, or possibly measure both the steam and air; the one method would then check the other. I have reasons to believe that if such a commission were appointed, they would not only discover the fact of the poor efficiency of the turbo blower, but also that the increased furnace efficiency expected from the turbo blower, due to steady blast, does not materialize and that the furnace works better with reciprocating blowing engines.

The reason for this is very easily explained when you consider that the air pressure is governed by the furnace. The amount of air delivered through the reciprocating blower will have to go through the furnace, no matter how much the pressure increases. With the turbo blower, however, it is different; its speed must correspond to the pressure against which it has to deliver. If the pressure tends to increase, due to the furnace, the turbo blower must speed up or it will churn part of the air. While the speed of the turbine can be regulated from the pressure, nevertheless there must be an increase in pressure first before there can be any force to act on the regulating mechanism of the turbine. The regulator cannot be made to act on too small pressure differences or it will cause a surging of pressure exactly the same as an engine governor that "hunts." If the regulation is effected by the volume, the effect will be exactly the same as above for the reason that a reduction in volume at a certain speed of the blower could only result from an increased delivery pressure.

J. E. Johnson, Jr., New York, N. Y.—Referring first to the question of the delivery of the turbo blower, at the February meeting of the Institute Mr. Valentine gave the results which he had obtained by the use of one of these blowers and claimed that his furnace had only required with its use 41.5 cu. ft. of air to burn 1 lb. of coke. I criticized this statement at the meeting as being less than practice indicated as at all probable. Professor Richards censured me for making this criticism,

but when I afterward intimated how strong were the grounds on which I made it, he calculated from Mr. Valentine's figures the amount of air which must have been blown, using the weights of materials charged and the gas analysis, and wrote me that I was entirely correct and that Mr. Valentine must have been using 15 per cent. more wind than the amount he stated, which was taken from the indicator of the turbo blower. This seems to me to show conclusively, especially when taken in conjunction with other data along the same line, that the calibration of this turbo blower was in error by a large percentage and that it was delivering considerably more wind than it took credit for. This error follows through all Mr. Rice's calculations. He has assumed that it takes 25 or 30 per cent. more air with the reciprocating engine, measured by piston displacement, than with the turbo blower and he has based all his figures for power consumed in the two cases upon this assumption. He is probably 15 per cent. too liberal to the turbo blower and not liberal enough by 5 or 10 per cent., on the basis of volumetric efficiency, to the reciprocating engine.

In regard to the steam engine, it seems to have been forgotten that in the country of the blind the one-eyed man is king, and that the turbo blower with its economical steam end may easily distance old, badly designed and perhaps worn-out blowing engines, and yet not equal modern ones in good condition. I, myself, have run an old engine at 19 revolutions with its outlet valve tightly shut, and there is no doubt that a turbo blower would beat this performance, particularly as its steam consumption per indicated horsepower hour was not less than 50 lb.; but the steam engine has not stood still and a reciprocating blowing engine of good, modern design, working under the conditions specified by Mr. Rice for the turbo blower, 200 lb. steam, 200° superheat, 28½ in. vacuum, of modern size and at a reasonable piston speed, will deliver an indicated horsepower for from 8½ to 9½ lb. of steam. The mechanical efficiency of a reciprocating compressor is probably higher than that of any other reciprocating apparatus because of the way in which the inertia of the reciprocating parts can be used to balance the changes in pressure in the steam and air ends. Ninety-five per cent. mechanical efficiency has been done by such engines and I have in my possession cards from one such engine showing 92 per cent. The volumetric efficiency is about 96 per cent. and the diagram efficiency about as much. The net result of these figures is that an air horsepower can be delivered by such an engine with about 10.4 to 11 lb. of steam, whereas the turbo blower on Mr. Rice's own showing requires 13.35 lb. The much smaller amount of power required, some 2,100 hp. as against the 2,500 required by the turbo blower, offsets the slightly higher cost of the reciprocating engine and gives a plant complete for just about the same expenditure as is required for the turbo blower, leaving the much smaller steam con-

sumption of the reciprocating engine as a net advantage. In addition to the smaller boiler power required for the reciprocating plant, there is the further advantage of the very much smaller condenser, since the engine will get along almost as well on 26 in. vacuum as on 28½, and only requires a condenser one-third as large and about one-third as much condensing water to produce it. On the other hand, it is a matter of common knowledge that the turbine loses efficiency very rapidly as the vacuum falls off. It is entirely impossible in three out of the four iron-producing districts in the country to obtain condensing water which will give a vacuum of 28½ in. for any considerable proportion of the year and as a result the possible economy of the turbo blower must be materially reduced.

Moreover, I am advised that a test has been made at the turbo-blower plant of the Iroquois Iron Co. and that this machine showed an efficiency of 55.6 per cent. against the 70 per cent. claimed for these machines by Mr. Rice. If the results of that test are reliable the advantage of the reciprocating engine is enormously increased.

Referring to Mr. Rice's final computations, he adds together the total operating cost of a plant of turbo blowers and turbo generators as against a complete gas-driven plant, thereby securing for the turbo blowers a part of the admitted advantage of the turbo generator. It may well be pointed out that the desirability of using a turbo generator should be discussed on its own merits and is entirely independent of the question of using the turbo blower. There is the best of reasons to anticipate that a plant composed of turbo generators and reciprocating blowing engines would prove superior to a plant made up exclusively of either turbine or reciprocating engines.

In regard to the fluctuations of pressure which Mr. Rice has shown, these seem to me to be of an order of size so small that it would be impossible to deduce their effect on the blast furnace. It is true that in many respects the blast furnace is a very delicate apparatus, but that its operation is affected to the extent which Mr. Rice claims by so small a fluctuation of pressure as he shows to exist, is scarcely within the bounds of probability. Such superfluous refinements are like tying a micrometer caliper to the end of a 50-ft. cotton tape. It is, therefore, impossible to admit the validity of Mr. Rice's claim of increased output, less coke, and less flue dust per ton of iron.

I would like to say that I believe that there is a field for the use of the turbo blower. I, myself, have some patents which I regard as valuable, covering its use for blast-furnace purposes along certain lines, and should be the last to say that there was no room for the turbo blower in the blast-furnace business, but it seems to me certain that its field is narrower than that assigned to it by Mr. Rice. Where economy is a matter of importance, the turbo blower should only be adopted at new

plants for reasons independent of economy, such as its small size and the small foundation it requires and its freedom from liability to cause pulsations to the earth and consequent vibration to other structures when installed on soft ground.

On the other hand, the turbo blower can be used to enormous advantage in reinforcing existing plants; using a low-pressure turbine, driven by exhaust steam from the existing blowing engines and driving a single-stage turbo, which pre-compresses the air, and delivers it to the inlet of the air cylinders at 6 or 7 lb. pressure. This greatly increases the pressure and volume which the engines can blow, increases the steam economy, and reduces the stresses in the engines.

Karl Nibecker, Youngstown, Ohio.—The data as given are most interesting and instructive, but several values mentioned do not appear to check with the results we have encountered.

While the actual operating conditions under which we are working may be somewhat different from those encountered by Mr. Rice, I do not feel that they are vitally at variance with some of the practice which he has probably encountered. In order that the information may be of service in general, we beg leave to call attention to some points at which we have found differences from the conditions as mentioned.

On p. 105, Mr. Rice speaks of the air delivered by the blowing engine as often being only 70 or 80 per cent. of the displacement of the piston. While this may be true in some cases, I do not feel that with a modern engine, in a reasonable state of repair, there is any justification for the volumetric efficiency being less than 90 per cent. The only condition under which I have encountered the low volumetric efficiency mentioned by Mr. Rice is in very old engines or engines which have not been given proper attention.

On p. 110 he states that the superior steadiness of the blast from the turbo blower results in less loss in fuel and ore, being evidenced by the reduction in the amount of flue dust. While this may be the case under certain conditions, it must not be taken too literally. When a furnace is blown with a turbo blower it is probably easier to drive the furnace somewhat harder and the increase of production obtained will probably be attended with an increased production of flue dust and a consequent reduction in yield. Our experience with the turbo blower would tend to carry out this assumption. The amount of dust produced would also seem to be dependent upon the kind of ore used. From our practice with Mesabi ores, we are inclined to believe that more dust may be produced by the blower than by the engine under the same conditions of working and production.

On p. 118 Mr. Rice gives a value of 65 cu. ft. per minute as the engine displacement per pound of coke. The term cubic feet per minute is

probably a typographical error, as the value should doubtless be given as 65 cu. ft. per pound coke. We believe that 53.5 would probably be a better value for modern practice. In case even as low a volumetric efficiency as 90 per cent. is assumed, 55 cu. ft. displacement per pound of coke should certainly be ample. We have repeatedly obtained indicator cards from an engine blowing a blast furnace producing an average production of better than 450 tons per day and found the indicated horsepower to be not in excess of 2,200 hp. In order to check the amount of displaced air per pound of coke with this horsepower, the calculation should appear as follows; we then have for the wind blown and horsepower developed, using 53.5 cu. ft. of air per pound of coke:

$$\frac{450 \times 2{,}240 \times 53.5}{24 \times 60} = 37{,}500 \text{ cu. ft. per minute}$$

$$\frac{5 \times 375}{0.94 \times 0.92} = 2{,}170 \text{ shaft horsepower.}$$

This checks well with tests made on blowing engines operating under conditions similar to the assumptions made in the paper.

The figure of 53.5 cu. ft. displacement per pound of coke, except in rare cases, would seem to be a very fair value and the above calculation checks with our practice.

In assuming 500° F. for the temperature of the gas delivered to the boilers, we feel that a very high value has been taken. Allowing 200° loss in radiation in piping and gas cleaning, this would require a furnace-top temperature of 700° F., and the gas must either be dry cleaned or not cleaned at all. If no cleaning is done, it would be practically impossible to maintain an efficiency of 70 per cent. in the boilers. We will, however, assume that the temperature of 700° F. is possible.

The gas required for boilers for reciprocating blowing engines would be as follows:

$$\frac{2{,}170 \times 14.15 \times 1{,}111.5}{103 \times 0.70} = 473{,}000 \text{ cu. ft. gas per hour.}$$

If the above value of 2,170 shaft horsepower is used for the gas engine the gas per hour required would be:

$$\frac{2{,}545}{0.21} \times \frac{2{,}170}{95} = 277{,}000 \text{ cu. ft. gas per hour.}$$

The assumption of steam conditions, efficiency, etc., made in calculating the gas per hour required by the turbo blower is good power-plant practice. I believe, however, that most blast-furnace plants would hesitate to install equipment for this pressure and superheat.

If the blower is figured on 150 lb. steam pressure and 28 in. vacuum, which I believe is more in accordance with present blast-furnace practice, the calculation appears as follows:

Assume 12 lb. of steam per shaft horsepower.

$$\frac{5 \times 350}{0.70} = 2{,}500 \text{ shaft horsepower.}$$

The heat per pound of steam is determined as follows:

Initial total heat	1,194.6
Final heat of liquid	69.1
	1,125.5

$$\frac{2{,}500 \times 12 \times 1{,}125.5}{103 \times 0.70} = 468{,}000.$$

Summarizing the gas requirements of the installations in cubic feet per hour, we have:

Reciprocating Steam Engine	Reciprocating Gas Engine	Turbo Blower
473,000	277,000	468,000

In the summary, the turbo blower shows a slight advantage over the reciprocating steam engine in gas consumption. This saving would be increased, due to the higher charges of investment, maintenance, depreciation, and attendance made against the engine. This difference in gas consumption may be very easily entirely eliminated by the excess dust produced when using Northern ores.

In conclusion, I would like to ask Mr. Rice how changes of atmospheric temperature and pressure affect the regulation of the machine; *i.e.*, if the temperature or pressure of the air changes and the governor setting is the same, will the amount of free air delivered by the blower vary more than that delivered by the engine under similar conditions?

Considering the governing device as a restricted area having the usual thermodynamic formula of velocities,

$$V = C\left(\frac{p_1}{\gamma_1}\right)^{\frac{1}{2}}\left[\frac{2gk}{k-1}\,\frac{1-\left(\frac{p_2}{p_1}\right)^{\frac{k-1}{k}}}{1-\left(\frac{F_2}{F_1}\right)^2\left(\frac{p_2}{p_1}\right)^{\frac{2}{k}}}\right]^{\frac{1}{2}}$$

and the ordinary formula for impact being

$$E = \frac{Mv^2}{2}$$

the impact by the disk becomes

$$E = CF_2\left(\frac{p_1}{\gamma_1}\right)\left(\frac{p_2}{p_1}\right)^{\frac{1}{k}}\left(p_1\gamma_1\right)^{\frac{1}{2}}\left[\frac{2gk}{k-1}\,\frac{1-\left(\frac{p_2}{p_1}\right)^{\frac{k-1}{k}}}{1-\left(\frac{F_2}{F_1}\right)^2\left(\frac{p_2}{p_1}\right)^{\frac{2}{k}}}\right]^{\frac{3}{2}}$$

In which,

E = Energy
C = Constant
F_1 = Area in square feet of upstream section
F_2 = Area in square feet of disk
p_1 = Absolute pressure at upstream section
p_2 = Absolute pressure at disk
γ_1 = Weight of 1 cu. ft. of gas at upstream section.

It then appears as though the effect of temperature and pressure changes may be exceedingly complicated and quite different from what might be expected from a superficial study of the governor.

W. Trinks, Pittsburgh, Pa.—Mr. Rice's paper on turbo blowers is doubtless interesting, but as I view it, mining engineers are particularly interested in machinery concerning mining operations directly, and for this reason I shall not only speak about turbo blowers, but also about turbo compressors.

On a trip which I recently made through England I found that one of the foremost engineering firms of Great Britain had built a number of turbo blowers for blast furnaces, but had built only a disappointingly small number of turbo compressors for mining purposes, whereas that same firm was a very extensive builder of reciprocating compressors for mines. Upon inquiry as to what caused the difference between these two products, and what made the turbo blower applicable in England when so few were used elsewhere, I was informed that the turbo blower can be used in England for blast furnaces on account of the following facts:

(1) The ores are lumpy and open.
(2) Coal is very cheap.
(3) The blast furnaces are low.
(4) The rate of driving the blast furnace is very slow compared to American practice, so that the pressure is low.

On the other hand, with air compressors the pressure is high, fuel prices are high at most mines except at coal mines, and last, the demand for air is irregular. It is a well-known fact that turbo blowers, from a certain percentage of rated delivery downward, show heavy pulsations and surging, which make the operation irregular and uneconomical.

Originally I had intended to limit my discussion to the foregoing remarks, but statements made here by other parties compel me to add to what I said before. First, I wish to speak about the statement which was made concerning the cost of installation of the turbo blower being so much lower than that of the blowing engine that the bankers would compel engineers to give preference to the turbo blower. With this I disagree, because I am convinced that any comparison leading to the

above given conclusion is made on the basis of engines as they were built 10 to 15 years ago. Engines built at that time run even now at a piston speed of 300 ft. per minute, whereas blowing engines built within the last five years are running at piston speeds of 600 to 800 ft. per minute. When it is remembered that the cost of a blowing engine per unit of air delivered is almost inversely proportional to the piston speed, it is evident that a modern blowing engine costs a great deal less than the old-time slow-speed blowing engine. This is information for the banker. Furthermore, blowing engines are more efficient now than they were formerly. Blowing engines with mechanically operated valves get out of adjustment, as may be proved by the fact that in one case with which I am familiar three such blowing engines which ran at 30 rev. per minute produced no more wind than one modern blowing engine of the same size running at 72 rev. per minute. Evidently the cost of wind production from the standpoint of first cost only had in this case been divided by three, although only reciprocating blowers were employed in the plant.

The statement was made that blowing equipment would have to pass through the same course of events as power-generating equipment; that is to say, turbo machinery would take the place of reciprocating machinery. I disagree with this statement also because there are two fundamental physical differences between the steam turbine and the turbo blower. The first one is that turbo machinery, such as a steam turbine, is most efficient at pressures below the atmosphere, and that the engine is more efficient at pressures above the atmosphere. Now, in the turbo blower the pressure range below the atmosphere is absent; that is to say, the blower must start to work at that pressure where it is less efficient than the reciprocating machine. The second reason for the lower efficiency of the turbo blower lies in the great difference between the conversion of pressure into velocity and the conversion of velocity into pressure. The steam turbine converts pressure into velocity, which is a highly efficient process. The turbo blower attempts to convert velocity into pressure and this is a very different proposition, as most of us know who have tried it.

The question of wear of the reciprocating engine compared to that of the turbo blower has been brought up. While I have not had personal experience with turbo blowers, I have had a great deal of experience with centrifugal pumps, and find that to throw the same quantity of water against the same pressure the speed must be gradually increased, because the leakage through the wearing rings increases with time. It is possible that the same leakage may occur in turbo blowers.

Finally, I wish to point out the great difference of opinion which exists between different engineers on the question of blowing equipment for furnaces. Whenever Mr. West or Mr. Freyn reads a paper, he proves conclusively that the gas engine is the only equipment that

can be considered for blowing purposes and that furnace owners cannot afford to install a turbo blower, even if it were given to them. On the other hand Mr. Rice proves just as conclusively that the turbo blower is the only right equipment and that gas engines cannot be considered in any event. From this standpoint, we should welcome the suggestion, which I believe was made by Mr. Iversen, that an unbiased committee should be appointed to investigate the relative merits of the two types of blowing machines.

Otto Banner, Phillipsburg, N. J.—There is practically no wear at all on the packing rings of turbo compressors or blowers; they stay unchanged just as the bearings of steam turbines do. They do not wear as they do in centrifugal pumps. In the case of one turbo compressor, these rings, after an eight years' run, were found to be fully as tight as when the compressor was started.

Turbo blowers in Germany are in use only on isolated and small blast-furnace plants, the tendency in that country being to prefer the gas-engine blower. If for a new machine steam has to be used, turbo blowers, and not reciprocating blowers, would be installed.

The fight between the gas and the steam engine is to a large extent decided by the cost of coal, but of late a remarkable development has taken place in Germany. They are installing in their newest plants gas-driven blowers, but steam-turbine driven electric generators. It would go too far to explain the reasons for this, but it is also very remarkable that in these new plants they install steam-turbine driven turbo blowers as reserves.

The turbo compressor for 85 to 100 lb. pressure has an undisputed field over there in the coal mines, first mixed-pressure and now live-steam turbines being chiefly used. The sizes of these units have been going up very rapidly during the last few years.

I advocate very strongly Mr. Iversen's proposition to nominate an independent commission to investigate exactly the merits of the reciprocating and the turbo blower. In Germany we have been up to this same controversy in the early days of the turbo compressor when it became necessary to find means for measuring the air input or output of these machines. I have been using low-pressure nozzles of varying sizes and the pressure difference used was usually not more than 20 in. of water column. The shape of these nozzles was that of a vena contracta; they have been adopted by the Society of German Engineers as standard for measuring actual delivered or taken-in air for reciprocating and turbo blowers and compressors, after a series of very exhaustive tests to find out the efficiency of these nozzles. For carrying out these tests, a big gasometer was used.

I wish these nozzles would be adopted in this country also because

their use would be fair to builder and user, and comparisons of all these different engines would be possible on the same basis of actually delivered air. Their use is extremely simple, as is also the figuring out of the test results. The Ingersoll-Rand Co. has been using these nozzles during the past few years on its reciprocating and turbo compressors and blowers. We have found, for instance, that on a 100-lb. reciprocating compressor the actually discharged volume reduced to intake conditions was 89 per cent., the indicator card showing 93 per cent.

J. E. Johnson, Jr.—My remarks were the result of an absolutely impartial discussion. I have no connection with any manufacturer of engines or turbo blowers; what I said was the result of an investigation that was impartial. I have been interested in the subject for a number of years.

H. S. Braman, Youngstown, Ohio.—The papers have been somewhat disappointing, due to the fact that they have been dealing with the furnace man's side of the question. We carried on an experiment a short time back, which I would not want to go on record as calling absolutely correct, with two blast furnaces of exactly the same size, using the same percentage of Mesabi ores, and practically the same fuel, one equipped with a turbo blower, and the other with a reciprocating engine. The air necessary to make a ton of iron in one furnace was 4,362 lb. with the turbo blower; on the furnace with the reciprocating engine 4,062 lb. of air, a difference of 300 lb.

Another thing we noticed with the reciprocating engine is that the furnace makes dust only during times of high pressure, but on the turbo blower, the furnace makes dust all the time, high or low pressure.

J. E. Johnson, Jr.—The figures I gave as to steam consumptions, were based on tests made by mining companies, and made on an engine built by one of the best engine builders of the country, and anybody who desires to get an engine from that builder, can get one with a guarantee that it will give better steam consumptions than I have quoted, and that guarantee will be backed up by the whole capital of that corporation. There is no question about the validity of those figures.

Arthur G. McKee, Cleveland, Ohio.—Since the installation of the first turbo blowers, I have been trying with the resources at my command to find out the facts in regard to the advantages of turbo blowers versus steam-driven reciprocating blowers and gas-driven reciprocating blowers, and I am frank to admit that up to the present time I have not been able to get anything which is conclusive and satisfactory in my mind in regard to this matter.

The discussion this morning has been somewhat partisan, which is bound to be the case in a large measure in discussions of this sort. How-

ever, there are some of us who are absolutely disinterested. We want to find out what is the best blower for the different conditions, regarding possible vacuums, fuel cost, and a number of other items which enter into this matter in a vital way, and I hope that something will be done by the members of the Institute with this in view, that we may get accurate non-partisan information about this whole matter. It is of great importance to the owners and builders of blast furnaces to know what is the best installation under each combination of conditions, taking into consideration the fuel cost, the vacuum which is available, the depreciation cost, the operating cost, and not overlooking the first cost, which is in some cases more or less lost sight of, but is of very great importance in providing blowing equipment.

Richard H. Rice (communication to the Secretary*).—The critics of the paper all base their remarks on two fundamental errors. The first of these consists in the ignoring altogether of *volumetric efficiency* as applied to a blast-furnace blowing engine, or in attempting to measure it from an indicator card of the blowing cylinder. The second is in considering all turbo blowers as alike in operative characteristics.

As to the first error, it arises in a confusion of definition. The term "volumetric efficiency" as used by me means the ratio of air actually discharged from the engine to the displacement swept through by the pistons, expressed in terms of cubic feet of free air at an inlet temperature of 60° F. P. 105 of the paper sets forth clearly the sources of loss of this efficiency and a slight study of these will be sufficient to show the impossibility of measuring this efficiency by an indicator card. The only way is by actual test as therein stated.

On p. 115 of the paper the difficulties of measuring the discharge from a reciprocating blowing engine are set forth. Since the paper was presented F. G. Cutler has read a valuable paper dealing with tests made on reciprocating blowing tubs to obtain the volumetric efficiency (American Iron and Steel Institute, Birmingham meeting, October, 1914), in which he shows that, using his method of throttling the discharge to reduce the pulsations of the flow, the engines tested have a volumetric efficiency of about 80 per cent. I believe that Mr. Cutler did not fully succeed in eliminating the irregularities of the flow, and that if anything this efficiency is too high. Since then, tests that I have made comparing turbo blowers and the latest design of reciprocating blowing tubs indicate about the same figure which Mr. Cutler gives as the value of this efficiency. This being so, the figures of Mr. Iversen are quite incorrect.

Mr. Johnson's speculations as to what would happen if a certain

*Received Mar. 8, 1915.

type of reciprocating steam engine having a certain assumed steam consumption (which I believe is incorrectly stated) were put to drive a blowing engine, are interesting; but such an engine, by reason of complicated mechanism, small cylinder clearance spaces, etc., is not adapted for the rough service of blast-furnace power plants.

Mr. Johnson's statement that a test has been made at the turbo-blower plant of the Iroquois Iron Co. is, I am informed, incorrect.

If it be inadmissible to study the details of blast-furnace operation in order to determine ways of improving the practice, then my tests on fluctuation of blast pressure need not be considered. It is, however, by just such studies that advance in the art must be made.

Among those who fall into the second error which I mentioned are Messrs. Nibecker and Braman. Discussing the operation of an apparatus not fitted with an adequate air governor, they reach the conclusion that the turbo blower makes more dust and produces less iron. Both these effects are precisely those which would result from overblowing the furnace and I have no doubt that this is what has been done.

Reports from the furnaces blown by the apparatus described in the paper indicate an amount of dust produced rather less than normal and an increase in the production, effects which are exactly opposite to those reported by Messrs. Nibecker and Braman.

The Iron Industry in Brazil

BY E. C. HARDER, MILWAUKEE, WIS.

(Pittsburgh Meeting, October, 1914)

Introduction

Few mineral deposits have in recent years attracted such general and widespread attention as the Brazilian iron-ore deposits, due mainly to the quantities of rich ore occurring here, in contrast to the ever-decreasing grade of ores shipped from many large producing iron-ore districts of the world. The general average grade of the iron ores shipped from the Lake Superior district[1] has decreased to approximately 50 per cent., natural, in recent years. The Rubio ore of Bilbao, Spain,[2] in 1890 averaged 55.5 per cent. metallic iron, dried; in 1902, 52.8 per cent., while at the present time a large proportion of the ores shipped from Bilbao is obtained by reworking the old dumps. In northern Sweden, where large magnetite deposits have only in recent years been extensively developed, selective mining is even now necessary in order to obtain high-grade Bessemer ores. During the last two years the iron ores of Chile have attracted much attention on account of the activities of the Bethlehem Steel Co. and others in the Chilean iron-ore region. Many of these ores are of good grade, yielding 67 and even 68 per cent. metallic iron, but judging from the writer's observations in this region a large proportion of them will be found to be of non-Bessemer grade.

The most important variety of iron ore in Brazil is hard, dense to specular hematite, occurring in Minas Geraes. A large number (89) of analyses of this type of ore have been averaged,[3] giving the following composition:

	Per Cent.
Iron	69.65
Phosphorus	0.0125
Silica	0.24
Combined H_2O	0.38

[1] Personal communication from Prof. C. K. Leith.

[2] Whitwell, William: Presidential address, *Journal of the Iron and Steel Institute*, vol. lix, pp. 24 to 50 (No. I, 1901).

[3] Communicated to the writer by H. K. Shearer.

In very few places in the world has iron ore been found in marketable quantities which even approaches this ore in grade, while in Brazil several hundred million tons of such ore are in sight, occurring practically on the surface, while besides this rich ore there is an almost incredible amount of lower-grade ores, *i.e.*, ores of 60 per cent. metallic iron and over. It may be found impracticable, when mining operations com-

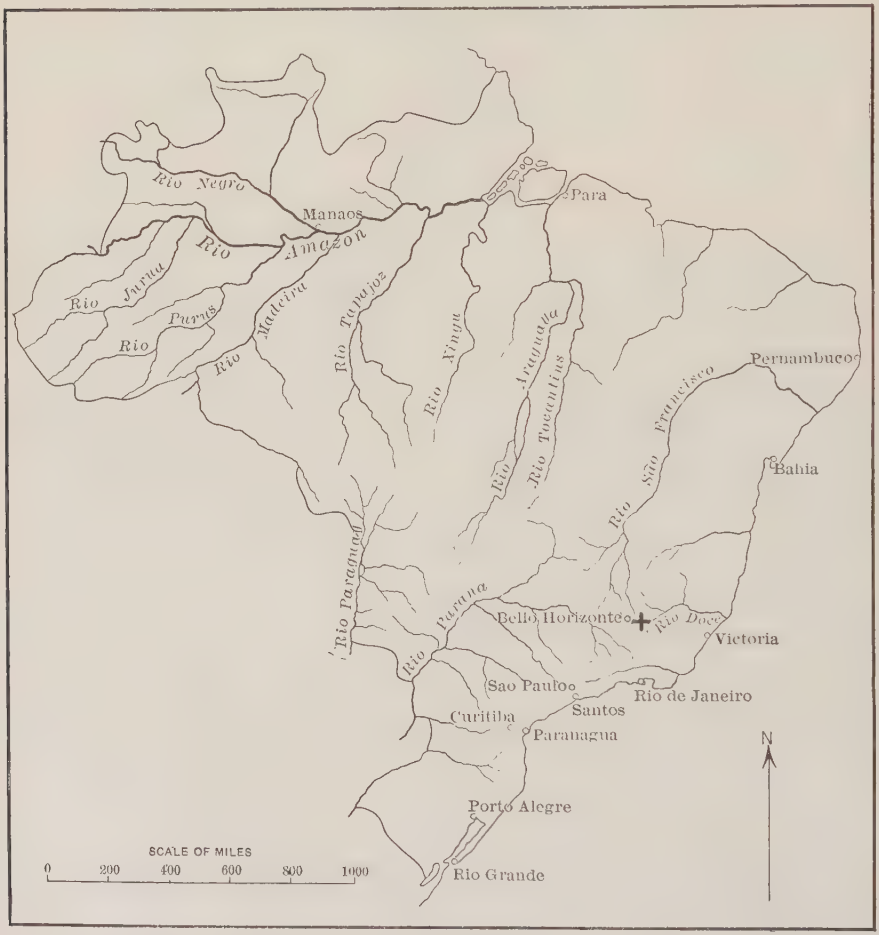


FIG. 1.—MAP OF BRAZIL, SHOWING LOCATION OF MINAS GERAES IRON-ORE DISTRICT.

mence, to mine exclusively the high-grade ores, and that admixtures of other ores will somewhat lower the general quality of marketed material; still it is safe to say that Brazil will be able to furnish for many years ores of Bessemer quality which will average more than 68 per cent. metallic iron.

Up to the present time no iron ore has been exported from Brazil, and

only a very insignificant quantity has been used to supply the small domestic furnaces. Plans have been made to ship a small quantity of ore from one of the mines in Minas Geraes during 1915, but it will probably be several years before any important quantities are exported. Conditions are not yet ripe in Brazil for the establishment of an iron-smelting industry of importance within the country. As there is no coal at present known in Brazil fit for smelting purposes, coal would have to be imported from Europe or North America, and since Brazil cannot afford a market for any great quantity of manufactured articles of iron, it would be necessary to export the greater part of the iron to other countries, thus necessitating double transportation, the bringing in of fuel and the taking out of manufactured iron. For this reason it is proposed to export the iron ore to foreign smelters where the fuel supply and the markets for the manufactured products are close at hand. Eventually, as the demand for iron in South America increases, smelting plants will doubtless be established in Brazil to supply the local market.

History of the Iron Industry

The history of the Brazilian iron industry is centered around two regions: (1) Ipanema, in the State of São Paulo, where small deposits of titaniferous magnetite have been operated intermittently since the beginning of the seventeenth century; and (2) Minas Geraes, where the extensive bodies of hematite occur which now seem destined to play an important part in the iron-ore industry of the world.

Dr. Derby[4] gives the following interesting history of the iron industry in São Paulo:

"The colonial records of Brazil register the fact that about 1590 an exploring party that set out from the town of São Paulo, founded about 40 years before, reported the finding of iron ore in a mountain situated 100 kilometers to the southward. Gold and silver were also reported from the same region, and acting on this information the Portugese Government took measures to promote the mining industry in the colony by sending out, in 1597, officials especially charged with this mission. The inclusion of an iron founder in the party indicates a special interest in the discovery of iron ore. One or two small forges were set up which commenced to produce iron probably about the year 1600, and continued in activity to about 1629. The place subsequently took the name of Ipanema which has ever since been inseparably connected with the long, though not brilliant, history of the iron industry in Brazil. There is a reasonable probability that the iron produced here was the first to be manufactured on the American continent.

"About the same time a forge was established close to the town of São Paulo in order to work the argillaceous ore that abounds in the vicinity, but it does not seem to have had a prolonged existence. . . . . .

In 1765 the production of iron was resumed at Ipanema in São Paulo, but again

[4] Derby, O. A.: The Iron Ores of Brazil, *Iron Ore Resources of the World*, p. 813 (Stockholm, 1910).

abandoned after a few years. An attempt made in 1800 to revive the industry, this time by means of a high furnace, was unsuccessful, and in 1810 a Swedish metallurgist, under contract with the Government, constructed four direct-process furnaces which continued in operation until 1818, when two high furnaces constructed by the German engineer officer, Frederic von Varnhagen, then in the service of the Portugese Government, were put into operation. These continued in blast, under Government administration and with a daily production of 3 to 4 tons until 1895."

In Minas Geraes we have the first definite information of the production of iron about the beginning of the nineteenth century, although it is reasonable to suppose iron was manufactured previous to this date by primitive African methods, with which many of the slaves imported for the purpose of working the gold mines in this region were probably familiar.[5] Dr. Derby[6] states that when Eschwege[7] arrived in the district in 1811 most of the smithies then existing produced their own iron, either in a small furnace constructed for the purpose, or by the spoonful in the ordinary blacksmith's forge.

Toward the end of 1812 a company formed under the direction of Eschwege began to produce iron at the rate of about 1 cwt. per day in a direct-process plant erected near Ouro Preto. After this the process spread rapidly, and before long the entire gold-mining district of Minas Geraes was dotted with little iron furnaces. In 1853, John Monlevade[8] a French engineer, then operating in Minas Geraes, estimated that 84 plants producing iron were located in the region between Ouro Preto and Itabira de Matto Dentro. These employed at least 2,000 persons, free and captive, and produced from 145,000 to 150,000 *arrobas* (2,175 to 2,250 tons) of iron yearly. Derby states that the total number of plants in the district in 1864 was estimated at 120.

Blast-furnace operations have not been extensive in Minas Geraes. Dr. Derby[9] states that an attempt was made to erect a high furnace at Morro de Pilar between 1809 and 1814 but it was unsuccessful.

In 1888, a charcoal furnace about 30 ft. in height and capable of producing about 5 tons of pig iron per day was erected at Esperanca[10] near Itabira do Campo on the Central Railroad of Brazil, and a few years later a similar furnace with a foundry in connection was built at Miguel Burnier, some distance south on the same railway. The Miguel Burnier furnace has been idle for many years but the foundry has been operating

[5] Derby, O. A.: *Op. cit.*

[6] Derby, O. A.: *Op. cit.*, p. 814.

[7] Baron W. L. von Eschwege, a Prussian military engineer, employed by the Portuguese government during the years 1810 to 1822 to develop the mining industry of Brazil. See *Trans.*, xxxiii, 407 (1902).

[8] Monlevade, João Antonio de: Letter to President of Province of Minas Geraes, Dec. 12, 1853.

[9] Derby, O. A.: *Op. cit.*

[10] Scott, H. K.: The Iron Ores of Brazil. *Journal of the Iron and Steel Institute*, vol. lxi, p. 248 (No. I, 1902).

with pig iron produced at Esperanca. The Esperanca furnace has been operated almost continuously to the present, using principally rubble ore obtained from the hills near Miguel Burnier. At the present time it is the only blast furnace in South America actively producing iron.

Direct Process of Iron Manufacture

There are at the present time about a dozen small plants in the Minas Geraes iron-ore district engaged in manufacturing iron from the ores by the direct process, that is, without the addition of fluxing material.

For smelting the ore by the direct process two types of furnaces are used: (1) the closed or crucible furnace, and (2) the open or Italian furnace.

Photo by H. K. Shearer

Fig. 2.—Serra do Caràca at Alegria. The ridge on the skyline consists of quartzite; the bare ridge in the middle background is iron formation.

While nearly all the plants have furnaces of both types, the crucible furnaces are more extensively used. These are generally arranged in series of four, six, or eight. A series of four furnaces will have about the following outside dimensions: 4 ft. high, 5 ft. wide and 10 ft. long. The opening or crucible itself is a vertical, cylindrical cavity about 2 to 3 ft. in depth and about a foot in diameter. From the base of the crucible an opening extends through the front wall through which the bloom is extracted after firing. In the rear wall of the crucible is a small opening which admits the blast, produced by water power.

The smelting operations consist in filling the crucible with successive layers of charcoal and crushed or pulverized iron ore, slightly dampened, then igniting and applying the blast. More ore and charcoal are gen-

erally added as the charge settles until a bloom of upward of 25 lb. is produced, the size depending more or less upon the size of the hammer with which it is to be worked. The bloom consists of a mixture of metal-

1. Caue. 2. Esmeril. 3. Conceicao. 4. Andrade. 5. Monlevade. 6. Morro Agudo. 7. Cocaes. 8. Bananal. 9. Morro da Mina. 10. Alegria. 11. Fabrica Nova. 12. Germano. 13. Timbopeba. 14. Capanema. 15. Mutuca. 16. Gongo Soco. 17. Gaya. 18. Aguas Claras. 19. Jangada. 20. Catta Branca. 21. Fabrica. 22. Casa de Pedra.

Fig. 3.—Map Showing the Location of the Principal Iron-Ore Deposits in Minas Geraes.

(Base adapted from a map by Gonzaga de Campos.)

lic iron, carbon, and slag. This mass, in a semi-molten condition, is pounded with a trip hammer, operated by an overshot water wheel, for the purpose of eliminating the slag. Very frequently it is necessary to reheat and

hammer a bar of bloom two or three times in order to make it fairly free from slag, the reheating being done in an ordinary forge.

The Italian furnace differs from the crucible furnace in being open, having only a floor and back wall. In the floor is a hollow or basin into which ore and charcoal are fed and in which the bloom is formed. The blast enters through the rear wall.

An ordinary direct-process furnace will produce about four bars of bloom daily, each 25 lb. and upward, or between 100 and 150 lb. of iron per day. One or two of the existing plants have about 20 furnaces, which makes the output of such a plant, when in continuous operation, nearly a ton of iron per day. As a rule, however, only a limited number of furnaces are operated at the same time.

The bars of bloom are later hammered into long, thin wrought-iron strips and these are manufactured into horse shoes and various agricultural implements, such as hoes, brush hooks, etc.

Fuel

Good fuel for smelting purposes is scarce in Brazil. Charcoal is usually made of poor wood which remains after the hardwoods and timber resistant to decay have been cut for building and other purposes. The wood remaining after this valuable timber has been removed is usually soft and yields a poor quality of charcoal, which serves the purpose for direct-process smelting but is unsatisfactory for blast-furnace operation.

Coal occurs in the States of Sao Paulo, Parana, and Santa Catharina in stratified rocks of probable Permo-Carboniferous age. It is said[11] to be of very poor quality, however, containing a large percentage of pyrite which makes it entirely unfit for smelting purposes. Other discoveries of coal are reported from the Amazon basin but these have not yet been proved to be of economic importance.

The Iron-Ore Deposits

The iron ores of Brazil are of two distinct types: (1) magnetite ores associated with igneous rocks occurring in the States of Minas Geraes, Sao Paulo, Parana, and Santa Catharina; and (2) hematite ores associated with metamorphosed sedimentary rocks in the State of Minas Geraes.

Magnetite Deposits

The magnetite ores are of relatively little importance, occurring only as small scattered deposits. The best known of these are found at Sao Joao Baptista, Minas Geraes; Ipanema and Jacupiranga, Sao Paulo; Antonina, Parana; and Joinville, Santa Catharina. Other deposits are

[11] Communicated to the writer by Dr. O. A. Derby.

reported elsewhere in Parana and Santa Catharina and also at localities in the States Rio de Janeiro, Minas Geraes, and Espirito Santo.

The Sao Joao Baptista deposits contain some magnetite of very high grade and are said to be extensive. They occur west of Oliveira station on the West of Minas Railroad in south-central Minas Geraes. A specimen of ore gave the following analysis.[12]

	Per Cent.
Fe	72.24
P	0.017
Siliceous matter	0.52
TiO_2	nil.

The Ipanema and Jacupiranga deposits consist of titaniferous magnetite occurring in pyroxenite associated with nephelinite and apatite-pyroxene rocks.[13] As is the case with most ores of this class every gradation exists between pure pyroxenite rock and iron ore. The content of titanic oxide in the Ipanema ore is said to be relatively low, but that in some of the Jacupiranga ores is said to be as high as 20 per cent. The Jacupiranga deposits are found in the southeastern corner of Sao Paulo near the sea coast, while those of Ipanema are located in the interior in the south-central part of the State.

The Antonina deposits occur in northeastern Parana close to the shore of a bay which extends northward from the port of Antonina. Eight or 10 different orebodies occur arranged at intervals along an approximately northeast-southwest line. The bodies are in the form of rough lenses inclosed in a basic igneous rock resembling diorite. The ore varies in grade; locally, masses of pure magnetite occur, but generally the ore consists of mixed magnetite and rock. A sample of fairly good-looking ore from one of the larger orebodies gave the following analysis.[14]

	Per Cent.
Fe	64.00
P	0.031
SiO_2	6.94
H_2O (combined)	0.63
Mn	0.20

Some of the Antonina orebodies consist of manganiferous iron ore, samples taken running as high as 25 per cent. manganese.[15]

The Joinville magnetite deposits are found about 16 miles southwest of Joinville and apparently consist of a number of scattered lenses in an

[12] Analysis by T. H. Lee, Servico Geologico e Mineralogico do Brazil, Rio de Janeiro.

[13] Derby, O. A.: On the Magnetite Ore Districts of Jacupiranga and Ipanema, Sao Paulo, Brazil, *American Journal of Science*, 3d ser., vol. xli, No. 244, p. 311, (Apr., 1891).

[14] Analysis by H. K. Shearer, Brazilian Iron & Steel Co.

[15] Analyses by T. H. Lee, Servico Geologico e Mineralogico do Brazil, Rio de Janeiro.

area underlain by granite and gneiss. The outcrops occur in the lower foot hills of the Serra do Mar, and are very near the coast, being on the edge of the belt of flat lowlands running parallel to the coast line. A sample taken over one of the outcrops gave the following analysis.[16]

	Per Cent.
Fe	71.30
P	0.041
SiO_2	0.45
H_2O (combined)	0.19
Mn	0.43

Hematite Deposits

The hematite deposits associated with the metamorphosed sedimentary rocks in Minas Geraes are the only known iron ores in Brazil to which any considerable importance is attached. They occupy a large area in the south-central part of Minas Geraes, being practically coextensive with the sedimentary formations in this region. The area within which the principal deposits are found is roughly 100 miles long from northeast to southwest, and about 60 miles in width. Within this region the orebodies are more or less segregated at certain localities.

The sedimentary series in which the iron ores occur is of probable pre-Cambrian age and rests unconformably on a basement of igneous rocks consisting of gneiss, granite and various crystalline schists. The lowest member of the sedimentary series is a massive quartzite of great thickness which is overlain by a thin discontinuous argillaceous schist. Above this schist is the remarkable Minas Geraes iron formation, which consists largely of a laminated, iron oxide-bearing quartzite known as itabirite, but which contains also beds of ferruginous schist and layers and lenses of iron ore. The iron formation is overlain by an extensive formation of schist and quartzite which contains iron formation and limestone beds in the lower part and which grades up into a formation consisting of quartzite with some schist, the highest known member of the sedimentary series.

Itabirite, the principal constituent of the iron formation, generally consists of alternating layers of quartz sand and iron oxide, but locally it consists of a granular mixture of quartz sand and iron oxide. It probably makes up more than 95 per cent. of the iron formation, the remaining 5 per cent. consisting of ferruginous schist and iron ore. Itabirite rarely contains less than 30 per cent. metallic iron and from this it grades up, with diminishing quartz sand, into ore, the lower limit of which may be taken as 50 per cent. metallic iron. Occasionally itabirite, very low in iron, apparently grades down into ferruginous sandstone or quartzite.

[16] Analysis by H. K. Shearer.

The iron formation varies greatly in thickness, in places being less than 50 ft. and elsewhere more than 4,000 ft. thick. The iron ore and ferruginous schist occur interstratified with the itabirite as beds or lenses varying in length and thickness. The iron ore appears to be a true sedimentary formation,[17] laid down at the same time as the inclosing rocks and later metamorphosed with them. The iron-ore beds have the same strike and dip as the inclosing rocks and outcrop with them at the surface. In many places hard-ore layers form the tops of hills or ridges or form conspicuous cliffs along the hillsides, due to their greater resistance to erosive processes.

When the iron formation weathers at the surface a blanket consisting of a mixture of itabirite and ore fragments cemented by limonite is formed. This blanket may vary in thickness from a few inches to more than 50 ft., and spreads over the surface of the iron formation as well as over the surface of adjacent formations such as schist or granite. This is the iron-ore conglomerate or breccia termed *canga*.

The so-called "itabirite ores" of Brazil are therefore of two distinct classes: (1) those occurring as original beds or lenses in the iron formation and (2) those resulting from the weathering and concentration of the iron formation. They may be classed as follows: Original or bedded ores: (1) hard massive ore; (2) soft powdery ore; (3) laminated or thin-bedded ore. Concentration ores: (1) canga (iron-ore conglomerate); (2) stream sand and gravel ores; (3) rubble ore; (4) enriched itabirite; (5) leached carbonate.

Of the above classes the bedded ores are by far the most important, especially the hard massive ore. This is of high grade and is the only type of ore being considered at present in plans dealing with the development of the Brazilian iron industry. The soft powdery ore is also of high grade, but on account of its consistency it is at present considered undesirable. The laminated ore is of lower grade than either of the other types of bedded ore. However, there occur in different parts of the district enormous deposits of this ore which will ultimately be of great importance.

Of the concentration ores the canga is the only type of importance. While the canga blanket covers many square kilometers, its grade is such that it cannot be considered desirable for the iron industry for many years to come.

Below is given the average range in iron and phosphorus content of the principal types of ore, assuming 50 per cent. metallic iron as the lower limit for iron ore. These figures are based on extensive sampling, much of it under the direction of the writer.

[17] Leith, C. K., and Harder, E. C.: The Hematite Ores of Brazil and a Comparison with the Hematite Ores of Lake Superior, *Economic Geology*, vol. vi, No.7, pp. 670 to 686 (Oct.-Nov., 1911).

Harder, E. C.: The Itabirite Iron Ores of Brazil, *Economic Geology*, vol. ix, No. 2, pp. 101 to 111 (March, 1914).

	Iron, Per Cent.	Phosphorus, Per Cent.
Hard massive ore	69 to 70	0.003 to 0.020
Soft powdery ore	50 to 69½	0.004 to 0.05
Laminated ore	50 to 68	0.003 to 0.07
Laminated ore (hydrated)	63 to 67	0.05 to 0.3
Canga	50 to 65	0.1 to 0.3

The hard massive ore is hematite, generally dense or finely specular, but occasionally, where strongly metamorphosed, it is coarsely crystalline and has an admixture of magnetite. It occurs in beds varying up to more than 450 ft. in thickness and to more than two-thirds of a mile in length. Hard ore rarely contains more than 1 per cent. of silica and its metallic iron content is remarkably constant.

The soft powdery ore is specular hematite in a fine friable form, so that most of it when dried crumbles to dust which will pass through a 100-mesh screen. Hard ore and soft ore frequently occur intermixed with each other, lenses or irregular bunches of soft ore occurring in hard-ore beds or irregular masses of hard ore occurring in soft-ore deposits. Soft-ore deposits are in general much smaller and more irregular than hard-ore deposits. Both occur as lenses or beds interlayered with itabirite or laminated ore.

Laminated ore consists of earthy to specular hematite. It is friable and breaks up into thin plates parallel to the lamination. It is also quite porous, allowing free circulation of ground water, on account of which large portions of these deposits are hydrated, especially near the surface, and consist of a mixture of hematite and limonite. Unhydrated, laminated and soft powdery ores grade from almost pure iron oxide, with little or no quartz sand, down to itabirite with more than 30 per cent. silica. The line between itabirite and ore in these cases is purely artificial. In the hydrated portions of the laminated-ore deposits the silica has generally, in large part, been removed, so that, although somewhat lowered in grade by the hydration, its silica content is generally small and the ore is fairly constant in grade.

Unhydrated ores of various types and even itabirite contain a very small percentage of phosphorus. It appears that phosphorus becomes concentrated during the hydration, either being brought in from the outside or remaining while other materials are being removed. Ore hydrated to beyond 2 per cent. combined water increases rapidly in phosphorus content. The hydrated portions of laminated-ore deposits rarely go to a greater depth than 50 or 75 ft. except along occasional more porous layers.

Surface weathering has but little effect on hard ore and soft powdery ore. The former is sometimes slightly hydrated along cracks, for a few inches from the surface, while the latter is generally altered to canga to a depth of 10 to 15 ft. and below this somewhat hydrated for an additional 4 or 5 ft. The laminated ore is affected to a greater depth by surface agencies, and along with the hydration there is removal of silica. This, however, rarely is sufficient to make an ore out of an itabirite, so that but a small percentage of this material can be classed with the concentration ores.

Canga consists of material derived by weathering from the iron formation. The canga blanket is generally thin along the tops of the iron-formation ridges and thick on the lower slopes, due to material being transported down the slopes and deposited near the base. Where iron-formation hills rise above the bordering schist or granite areas the canga blanket extends out over these areas. The best grade of canga occurs on the iron-formation areas, especially where ore occurs with the itabirite. Frequently such canga is made up, to a large extent, of irregular fragments of ore cemented by limonite and hydrated hematite. The canga overlying areas of other rocks is of lower grade than that overlying iron-formation areas, containing a large proportion of impurities such as clay (perhaps containing bauxite, etc.) and silica. The farther away from the iron formation the poorer is the canga, until finally it becomes merely an ocherous clay. Over and close to iron-formation areas the fragments in the canga are usually angular, farther away they are small and rounded and finally they disappear. Thus there is every gradation of canga, from ferruginous clay to iron ore with 65 per cent. metallic iron.

Of the other classes of concentration ores the only ones of importance are rubble ores and leached itabirite.

Rubble-ore deposits occur as talus under extensive outcrops of hard ore. They are simply masses of hard-ore fragments broken off from the outcrops, and are of the same grade as the bedded hard ore from which they are derived.

Where canga overlies compact thin-bedded itabirite it frequently retains the itabirite texture though most of the silica is removed. This canga *in situ*, exhibiting original textures, is rarely more than a few feet in thickness, being underlain by sandy itabirite. It is leached itabirite.

Stream sand and gravel consisting of hematite occur along some of the streams cutting through the iron-formation belts. They are rarely pure enough to be called ore.

Leached carbonate is an earthy material consisting of hydrated iron or manganese oxides, or a mixture of the two. It is derived from the decomposition of impure limestone beds occurring in the upper part of the iron formation and in the lower part of the overlying schist. It is seldom rich enough in iron to be called ore.

Transportation

In relation to such enormous quantities of high-grade iron ores as exist in Brazil, the principal problem is, of course, that of transportation. For more than 25 years the Central of Brazil Ry., owned and operated by the Brazilian government, has run northward from Rio de Janeiro through the center of the iron-ore district. At the present time it continues far to the north to Pirapora on Rio Sao Francisco, from which point it is possible to travel by steamboat and railway to Bahia. The Central of Brazil is a broad-gauge line (1.6 m.) from Rio de Janeiro as far as Miguel Burnier in the southern part of the iron-ore district, a distance of about 310 miles, and beyond this point it continues northward with a narrow gauge (1 m.). From this point also a branch line runs eastward to Ouro Preto and Marianna while another branch line runs eastward from Sabara to Caethe and Santa Barbara, both in the iron-ore district. The latter branch is being continued to Itabira de Matto Dentro. From General Carneiro, north of Sabara, a branch line runs southwestward to Bello Horizonte, the capital and principal city in Minas Geraes. A broad-gauge line is now being constructed from Lafayette, a station south of the iron-ore district, to Bello Horizonte by way of Rio Paraopeba, west of the iron-ore district. This will make a direct broad-gauge connection between Rio de Janeiro and Bello Horizonte.

The Leopoldina Ry. has a network of lines in the southeastern part of Minas Geraes and in the neighboring States of Espirito Santa and Rio de Janeiro. Saude, the terminus of one of its northernmost branches, lies about 20 to 25 miles east of the eastern border of the iron-ore district. The Leopoldina Ry. has a gauge of 1 m.

The West of Minas Ry. has a branch line of 1-m. gauge which runs into Bello Horizonte from the west. The main line of this road, however, is some distance west of the iron-ore district.

The Central of Brazil Ry. was not planned as an ore-carrying road. It crosses two divides between the iron-ore district and Rio de Janeiro, one south of Barbacena, known as Serra da Mantiqueira, and the other north of Rio de Janeiro, known as Serra dos Orgaos. Heavy grades and sharp curves abound over large portions of the line. The Brazilian government, however, has taken several contracts for hauling iron ore over this road to Rio de Janeiro from points in the western part of the iron-ore district at the rate of $1.50 per ton. Small shipments of manganese ore are being transported over the road at the present time, and judging from difficulties encountered in this traffic, it seems probable that much reconstruction will be necessary before this railroad can handle even a small fraction of the iron ores which must ultimately be transported from the district.

The Leopoldina Ry. and the West of Minas Ry., on account of their

sharp curves and prohibitive grades, are not likely to enter into the ore-carrying traffic.

It seems probable, therefore, that by far the greater part of the ore is destined to leave the district by another route than those mentioned, viz., by the Victoria to Minas Ry., which leaves the sea coast at Victoria and runs westward up the Rio Doce. This railroad, which is now under construction, will enter the iron-ore district from the northeast and will tap first those deposits which are now farthest from lines of transportation. The distance to Victoria from the eastern part of the iron-ore district over this route is about 375 miles, the entire distance being on a down grade following the course of the Rio Doce. About 275 miles of the line has been constructed, but the greater part of this will probably need reconstruction since it was not originally planned to make this an ore-carrying road.

In connection with the rapid development of the district it may be of interest to note that previous to 1910 most of the iron-ore deposits were in the hands of Brazilian owners, many of whom were descendants of persons to whom the original land grants were given more than 100 years ago for purposes of gold mining, while a few deposits were owned by the gold-mining companies operating in the district.

At the present time the principal deposits in the eastern part of the district, that is, in the part tributary to the Victoria to Minas Ry., are controlled by English, French, and American interests, who have done extensive exploration and development work. In the western part of the district some deposits are in the hands of Brazilians, as are also a few minor deposits in the eastern part, but here foreign interests, including the German, have made extensive purchases as well and have done much development work. It is from this part of the district that small shipments of ore are expected to be made in 1915 over the Central of Brazil Ry.

Labor and Wages

A serious difficulty to be contended with in extensive mining operations in Brazil, is that of labor. The wages are low, ranging from about 75c. to $1.25 per day for ordinary labor and about twice as much for skilled labor, but the difficulty is not only in obtaining laborers, but in keeping them. The gold mines in the district are in constant difficulty in this respect, and unsuccessful attempts have been made to import Spanish labor, while recently one of the mining companies imported a number of Japanese laborers. The native Brazilian laborer is averse to working underground and prefers agricultural work with lower wages to mining work. As the iron-mining operations will, however, be largely of the nature of quarrying, there should be less difficulty in obtaining and retaining laborers than is experienced in the gold mines.

Discussion

I. C. White, Morgantown, W. Va.—I have seen some of the great iron-ore deposits described in this paper. One of these in the State of Minas Geraes appears to be an immense mountain of iron ore about 2,000 ft. in height. All over its surface from bottom to top may be seen large masses of rich iron ore, varying in weight from a few pounds to several tons, and from all that one can judge without exploitation, the quantity of available ore is certainly very large in this particular mountain. Whether it will prove a mere surface deposit or shell, like the famous Iron Mountain of Missouri, is for future exploration to determine.

The question of transportation is the main factor in determining the availability for exportation of these Brazilian iron ores. There is no coal in Brazil that can be manufactured into coke without expensive previous treatment, since the best of the raw coal in the States of Rio Grande do Sul, Santa Catharina, and Parana, contains about 35 per cent. of ash, of which about 6 per cent. is sulphur, and coal of this composition cannot be utilized in iron smelting without purification.

When I was chief of the Brazilian Coal Commission in 1904 to 1906, I had a cargo of this coal shipped to Kalk, Germany, and treated in the great coal-testing plant of the Humboldt Engineering Works, located just across the Rhine from Cologne. It was found that by crushing and washing the Brazilian coal, it yielded 33 per cent. of briquetting coal, containing only 10 to 12 per cent. of ash (of which 1 per cent. was sulphur), and 42 per cent. of slack coal, with 25 per cent. of ash and low in sulphur, so that about 33 per cent. of the Brazilian coal could be converted into coke of fair quality, as shown by the following average of three analyses, using air-dried coal, of briquets made from the Barro Branco coal bed, State of Santa Catharina:

Moisture	1.56
Volatile matter	31.66
Fixed carbon	56.73
Ash	10.05
Total	100.00
Sulphur	1.34
Phosphorus	0.003
B.t.u	13,395

A briquet from the Sao Jeronymo coal, State of Rio Grande do Sul, gave the following results on analysis:

Moisture	5.03
Volatile matter	33.42
Fixed carbon	50.77
Ash	10.78
Total	100.00
Sulphur	0.61
Phosphorus	0.033
B.t.u	12,496

The briquets were, of course, made from crushed and washed Brazilian coal to which about 5 per cent. of pitch had been added as a binder. These coals when crushed and washed would yield a fairly good coke, but nevertheless the all-important question of transportation would remain to be solved, since the coal deposits are about 1,000 miles distant from the ore and limestone deposits and railway transportation in Brazil from the coast to the interior and *vice versa* is very expensive. The main reason for this is that a great wall of granite 2,000 to 3,000 ft. high parallels the coast from Bahia southward to Florianapolis or beyond, through which the rivers descend in cataracts, and thus have not yet graded out easily accessible railway routes from the seacoast to the interior, so that railways are compelled to climb up over this granite wall with steep grades, driving tunnels and making deep cuts and rock shelves along high cliffs at vast expense, the funds for which have been provided mostly by foreign capital at high interest rates. The scenery on the journey from Paranagua up to Curityba in the State of Parana cannot be excelled for rugged grandeur and wildness on any of the railways of Switzerland. The great forests of Parana, into which this railway enters, cannot compete, however, with lumber from Florida and Russia in the markets of the coastal cities of Brazil, although transported for thousands of miles instead of hundreds, owing to the transportation costs from the mountain region to the seacoast, a distance of less than 100 miles. Hence, whether it will be possible in the near future to bring the Brazilian ores 400 to 500 miles from the interior to the seaboard at a cost low enough to make their exportation to the United States or to Europe feasible is a question that has not yet been solved; and it certainly will not be in this generation, when good coal brings about $10 a ton by the shipload delivered at the seaport cities of Brazil, and $15 to $20 per ton when delivered at the ore mines of the interior of the country.

At one locality in the State of Minas Geraes, there occurs such an assemblage of minerals necessary in the manufacture of iron and steel as probably occurs nowhere else in the world; namely, a deposit of excellent limestone standing nearly vertical, upon one side of which is a deposit of good iron ore, many feet in thickness, and on the other side 20 ft. of good manganese, which is mined and shipped to the United States. It might be possible to ship coke from the United States to this

rich mineral zone of Brazil, and thus inaugurate a Brazilian iron and steel industry, since at the time of my travels there (1904 to 1906) only one small charcoal furnace making 5 to 6 tons a week was in existence in the entire area of that great republic.

E. C. Harder (communication to the Secretary*).—Replying to Dr. I. C. White's discussion. While some of the Brazilian iron-ore deposits are of great size it is only rarely that an entire mountain or hill is composed of ore. As the iron ores occur in beds and these beds are generally of considerable hardness they frequently form ridges, the shapes of which depend upon the attitude of the layers. Where the beds are gently inclined the dip slope of a ridge is usually gradual, while the opposite slope is steep and frequently precipitous, due to more rapid erosion of softer rocks underlying the capping bed of iron ore. In such a case the iron-ore bed usually extends down the dip slope for varying distances, depending upon its horizontal extent. It will also extend some distance down the precipitous slope, depending on the thickness of the bed, while below the iron ore the underlying rocks appear, such as itabirite or ferruginous schist. In many places these underlying rocks are concealed by a thick talus of rubble ore broken and fallen from the upper slope. The fragments forming the talus vary in size up to many tons.

In places where the bedding is vertical or nearly so hard iron-ore layers may form a sharp, steep-sided ridge, of which the ore forms the crest while the slopes are formed by softer adjacent rocks. In this case also the slopes are generally covered by a mantle of ore fragments.

The Brazilian deposits differ from those of Iron Mountain, Missouri, in that they are sedimentary beds or lenses like limestone or shale, while those of Iron Mountain are veins in igneous rocks. Naturally the Brazilian ores may be expected to extend to a greater depth down the dip, since sedimentary beds are in general much more continuous than veins.

With regard to transportation, the question is one of distance rather than high rates. The government railway (Central of Brazil Ry.) has taken contracts to transport iron ore from the ore district to Rio de Janeiro, a distance of 300 to 400 miles, for 6s. (about $1.50) per ton, not a high rate. The principal difficulty lies in the fact that most of the ore will have to be hauled this great distance by rail to reach the coast and then, after trans-shipment, will be subjected to about 5,000 miles of ocean transportation to reach North America or Europe. Thus only the very high-grade, hard ores would seem to warrant the expense of transportation at the present time.

As stated in the original paper, it seems improbable that the

* Received Dec. 7, 1914.

Central of Brazil Ry. on account of sharp curves and prohibitive grades, not taking rates into consideration, will be able to transport more than a small percentage of the ore which must eventually be taken from the district. For this reason another railroad, the Victoria to Minas, is being constructed, which, instead of going south to Rio de Janeiro, as the Central of Brazil Ry. does, goes in an easterly direction and reaches the ocean near Victoria, a small port 300 miles northeast of Rio de Janeiro. This railroad will follow the Rio Doce on a down grade for the entire distance to the ocean and no part of it will have a steeper grade than 1 per cent. Over a railroad thus constructed it should be possible to transport ore with a minimum of expense.

Dr. White suggests that it should be possible to bring coal to the iron-ore fields and smelt ore at a profit there, since both limestone and manganese ore are available right in the iron-ore district. While it is true that manganese ore occurs locally in the district and that in some places impure limestone occurs, these materials are after all of minor importance and the great necessary item is coal or coke. To ship coal to South America for the purpose of smelting iron ore and then returning the iron products to Europe seems wasteful and quite unwarranted. Even if some portion of the product were utilized in South America, why should it be more advantageous to bring the coal to the iron-ore fields than to take the iron ore to the coal fields and manufacturing centers? In the history of iron mining the natural trend has been for the ore to go to the coal fields and for manufacturing centers to be established there. This we see in case of the Spanish and Swedish ores which go to England and Germany for smelting. This also has been the history of the Lake Superior ores, which have gone, and in large part are still going, to the Pennsylvania coal region for smelting. It is only within recent years that coal is being taken West for iron-ore smelting purposes and that extensive smelting operations have been started at Gary and West Duluth. It has been found more profitable to transport iron ore to established manufacturing centers and to return the finished products to the original source than to manufacture these articles at the source. Thus in Brazil at the present time the limited number of articles of iron manufactured with cheap labor and from raw materials, such as wood and iron ore, which cost but little, find it difficult to compete with similar articles imported from Europe and North America.

The Brazilian iron-ore problem has been carefully studied during the last few years by men competent to judge, and the prevailing conclusion has been that the ore can be transported to Europe to be smelted and the products sold at a profit, whereas it would be quite out of the question to attempt the establishment of an extensive iron industry in Brazil.

Pig Steel from Ore in the Electric Furnace *

BY ROBERT M. KEENEY, PITTSBURGH, PA.

(New York Meeting, February, 1914)

CONTENTS

	Page
Introduction	161
Conditions Necessary for the Electric-Furnace Production of Pig Steel	163
Experiments on the Electric-Furnace Production of Pig Steel	165
Regulation of Carbon	166
Variation of Limestone and Coke in the Charge	167
Variation of Coke in the Charge	167
Variation of Limestone in the Charge	167
Variation of Fluorspar in the Charge	169
Impurities in the Pig Steel	171
Silicon	171
Phosphorus	172
Sulphur	174
Loss of Iron in the Slag	175
Adaptability of the Process to Continuous Operation	176
Cost of Production	178
Summary	179

Introduction

At the beginning of the use of the electric furnace, for the manufacture of calcium carbide and ferro-alloys, experimental work was conducted in it upon the production of steel from iron ore. Stassano performed two series of experiments in Italy, the first[1] during the years 1898 to 1902, and the second[2] in 1908.

For many years steel and wrought iron have been produced directly from ore on a small scale in the forge, crucible, and reverberatory furnaces; but the difficulties attending the operation of these combustion processes have prevented their adoption to any great extent. The combustion

* Published by permission of the Director of the U. S. Bureau of Mines.

[1] Stassano, E.: *Electrochemical and Metallurgical Industry*, vol. vi, No. 8, p. 315 (Aug., 1908).

[2] Catani, R.: *Journal of the Iron and Steel Institute*, vol. xxxiv (1911, No. II), p. 215; *Metallurgical and Chemical Engineering*, vol. ix, No. 12, p. 642 (Dec., 1911).

processes which have attempted to produce a product which might be classified as a pig steel rather than pig iron, have failed largely for economic reasons rather than because of metallurgical difficulties. They were not adapted to the large units necessary to turn out a large tonnage, as is the case in the manufacture of pig iron.

The introduction of the electric furnace into metallurgy somewhat renewed the interest of metallurgists in direct processes for the manufacture of steel. This is because the electric furnace can be operated with the proper metallurgical conditions for the production of pig steel rather than pig iron, and because in a possible design of shaft furnace similar to the blast furnace there is a greater chance for the production of a larger tonnage than is possible in the case of furnaces used in combustion direct steel processes. The early experiments of Stassano showed possibilities for a direct steel process in an electric furnace of the hearth type, which, however, had the same objectionable features as the combustion processes—a small output and a high cost of production. Later, in 1909, during the experiments[3] on the production of pig iron in the electric furnace at Domnarfvet, Sweden, 280 tons of crude iron containing from 0.95 to 3.09 per cent. carbon were produced. This showed the possibility of producing pig steel in an electric shaft furnace. Later at Trollhättan, Sweden, these results were confirmed in a larger furnace.[4]

In this paper the writer does not advocate the production of pig steel from ore in the electric furnace as a competitor of the present method of steel production with pig iron as an intermediate step, but presents the general metallurgical side of such a process and its economical possibilities in regions where power is cheap, and fuel and reducing materials are expensive, as, for example, Sweden, California, British Columbia, and the western coast of South America. The field of the electric furnace in pig-iron manufacture is restricted to such areas, and possibly will never have great application even in favorable districts, because of the cheapness with which iron ore can be delivered from western coast deposits to furnaces upon the eastern coast of the United States on the completion of the Panama Canal. But when in these countries local demand for pig iron and steel is great enough, the electric furnace has an opportunity. In such cases, if steel is the desired final product, it seems more feasible and cheaper to produce as pure a product as possible—*i.e.*, pig steel—for any subsequent refining operations, rather than pig iron. This would prove to be the case for two reasons:

1. As has been found to be the case at Heroult, Cal., Domnarfvet,

[3] Yngström, L.: Report on the Experiments Conducted at Domnarfvet, Sweden, on Smelting Iron Ore by the Electrothermic Process. *Bulletin No. 3, Department of Mines, Canada*, p. 19 (1910).

[4] Lyon, D. A.: The Use of Electric Furnace Pig Iron in the Open Hearth Furnace. *Metallurgical and Chemical Engineering*, vol. x, No. 9, p. 539 (Sept., 1912).

and Trollhättan, pig steel can be produced more cheaply in the electric shaft furnace than pig iron, because of the greater output per unit of electrical energy consumed.

2. Owing to the lower percentage of impurities in pig steel, 1 to 3 per cent., instead of 4 to 8 per cent. in pig iron, a greater output can be obtained from the final refining furnace at a lower cost.

Conditions Necessary for Electric-Furnace Production of Pig Steel

In his discussion of electric-furnace production of pig iron and pig steel[5] before the Engineers' Society of Western Pennsylvania, Dr. J. W. Richards defines pig steel as "a metal with 2.2 per cent. or less of carbon, a very small amount of silicon and manganese, low in sulphur and phosphorus, and made directly from iron ore in the electric pig iron furnace." That is, in chemical composition it is steel. If it resembles steel in chemical composition, there is no reason why it should not possess the same physical qualities as steel, when subjected to the same conditions of pouring and heat treatment as a combustion-furnace steel of the same composition with which it may be compared. Or it should conform to the definition of steel as given by Stoughton:[6] "iron which is malleable at least in some one range of temperature, and in addition is either (a) cast into an initially malleable mass; or (b) is capable of hardening greatly by sudden cooling; or (c) is both so cast and so capable of hardening."

A process to produce efficiently a metal conforming to these requirements should satisfy the following conditions:

1. As the product in order to be a steel must not contain over 2.2 per cent. carbon, it should be possible not only to keep the percentage of carbon in the metal below that amount, but it should also be possible to regulate the carbon within reasonable limits; *i.e.*, control the composition of the metal.

2. The product should contain percentages of silicon, phosphorus, and sulphur either below the limit set by consumers of Bessemer and open-hearth steels, or at least low enough not to require prolonged refining in another furnace.

3. The loss of iron in the slag should not be excessive.

4. The furnace used must be adapted to continuous operation and the production of a large tonnage.

[5] Richards, J. W.: Electric Furnace Production of Pig Iron and Pig Steel. *Proceedings of the Engineers' Society of Western Pennsylvania*, vol. 28, p. 83 (1912); *Metallurgical and Chemical Engineering*, vol. x, No. 7, p. 397 (July, 1912).

[6] Stoughton, B.: *The Metallurgy of Iron and Steel*, p. 7 (1908).

5. By the process it should be possible to produce pig steel at a greater profit than by existing methods.

Considering the possibility of the use of the electric furnace to satisfy these requirements:

1. As it is not necessary to introduce carbon for fuel in the electric furnace, it is not necessary to have excess carbon in the charge beyond that needed for reduction, therefore no difficulty should be experienced in keeping the carbon of the product below 2.2 per cent.; and it should be possible to regulate the carbon within reasonable limits.

2. The temperature of the electric furnace can be regulated by the power input, so that very basic slags can be used, which should result in slagging of silicon, phosphorus, and sulphur.

3. While the loss of iron in the slag with an electric furnace running on pig steel would probably be greater than in the case of pig iron, owing to the weaker reducing atmosphere of the pig-steel furnace, it should not be so excessive as to prohibit the use of such a process under favorable economic conditions.

4. The electric shaft furnace has been proved to be easily adapted to continuous operation for pig-iron production, and should operate more uniformly on pig steel than on pig iron because there is not the tendency for carbon to accumulate in the furnace.

5. In localities where electric-furnace pig iron can be produced at a profit, pig steel for further refining could be made at a profit if the market demand was for steel.

The direct production of steel from iron ore in the electric furnace has never passed beyond the laboratory stage. Just as pig iron can be produced from iron ore in the electric furnace, so can pig steel be produced by a close regulation of the carbon content of the charge. Sufficient carbon is provided for reduction of the ore, but not enough to combine to any appreciable extent with the product. This of course cannot be done in the blast furnace because of the excess carbon present for fuel. The ore is reduced to pure iron, but this combines at once with any free carbon present. In the electric furnace the electrical energy does the heating, while carbon simply performs the chemical function of reduction. The reduction of the ore to pig steel is comparatively simple, but some difficulty has been experienced in preventing a high loss of iron in the slag, due to the low carbon content of the charge. Also, as in all direct steel processes, the commercial success of a similar electric furnace process is yet to be determined by operations conducted on a large scale.

Some experimental work performed by the writer in 1911 under a Carnegie Research Scholarship of the Iron and Steel Institute of Great Britain, and the work of others, serve to show the metallurgical possibilities of the electric furnace for the production of pig steel from ore, and whether the process satisfies the five stated requirements.

Experiments on the Electric-Furnace Production of Pig Steel

Five groups of experiments were made on the electric-furnace production of pig steel, arranged as below:

Group I. A series in which both limestone and coke were varied in the charge.

Group II. A series in which the amount of coke in the charge was varied, other components remaining constant.

Group III. A series in which the amount of limestone in the charge was varied, other components remaining constant.

Group IV. A series in which a part of the limestone was replaced by varying amounts of fluorspar, other components remaining constant.

Group V. An experiment in which the furnace was operated continuously rather than intermittently, as in the other experiments.

Table I.—*Analyses of Raw Materials*

Hematite

		Per cent.
Fe_2O_3	(57.0 Fe)	81.40
SiO_2		9.25
Al_2O_3		4.30
CaO		1.00
MgO		0.40
MnO		0.15
Phosphorus		0.124
Sulphur		0.14

Coke

Fixed carbon	81.48
Volatile and combustible matter	0.25
Ash	17.92
Moisture	0.35
FeO	1.20
SiO_2	13.62
CaO	0.85
Phosphorus	0.08
Sulphur	0.54

Limestone

$CaCO_3$	(54.5 CaO)	97.50
MgO		trace
SiO_2		0.70
Al_2O_3		0.80
Phosphorus		0.008

Fluorspar

CaF_2	90.00
$CaCO_3$	0.80
Al_2O_3	8.50

Because of the impossibility of tapping a small furnace cleanly after each experiment, it seemed advisable, in order to more closely represent large-scale working, to calculate the theoretical weight of the pig steel present from the loss of iron in the slag and the analyses of the pig steels tapped, considering all iron available which did not pass into the slag. The results of these calculations were shown to check closely with the total amount of metal obtained during all of the experiments, including the amount cleaned out of the furnace at the end. Of the total calculated 89.4 per cent. was tapped, and 93.6 per cent. was reduced according to calculations.

These experiments were performed in a furnace, lined with magnesite, having two upper vertical graphite electrodes connected in parallel, and a conducting hearth of iron rods imbedded in magnesite, the hearth and the graphite electrodes being connected in series. The interior dimensions of the crucible were: length, 9 in.; width at the bottom, 4 in.; depth, 12 in. This furnace took from 10 to 12 kw. of single-phase current at 30 to 50 volts, and held about 25 lb. of cold charge.

Analyses of the raw materials used are given in Table I.

Regulation of Carbon

The pig steels produced varied in composition with the coke in the charge from a low-carbon to a high-carbon pig steel. With one exception, they all contained less than 1.75 per cent. This exception was a white iron containing 2.25 per cent. carbon. The percentage of manganese was uniform, with an average of 0.11 per cent. The silicon varied to a considerable extent with the basicity of the slag, but could be kept below 0.15 per cent. without difficulty The average percentage of silicon in the pig steel was 0.12 per cent. The percentages of phosphorus and sulphur in the pig steels varied under different conditions. The loss of iron in the slag was not excessive. From the results obtained, it seems that the regulation of carbon in a pig steel produced from ore in the electric furnace is possible within rather close limits.

As shown in Table IV, the percentage of carbon in the pig steel gradually increased with the increase of carbon in the charge, up to about 1.50 per cent. in the pig steel, above which the amount of carbon absorbed does not increase as rapidly as the increase of carbon in the charge. To determine whether there is any definite relation between the carbon charged and the carbon entering the steel, the results of some of the experiments have been figured to pounds of carbon charged per pound of pig steel reduced, and from this the amount of carbon entering the metal has been deducted in each case, giving the pounds of carbon actually used per pound of pig steel reduced for reduction only, Table II.

TABLE II.—*Carbon Regulation*

Experiment No.	Carbon in Steel	Lb. Carbon Charged per Lb. Pig Steel Reduced	Lb. Carbon Charged for Reduction per Lb. Pig Steel Reduced	Lb. CaO Charged per Lb. Pig Steel Reduced	Lb. CaF_2 Charged per Lb. Pig Steel Reduced
	Per Cent.				
3	0.29	0.2810	0.2781	0.6780	.....
5	0.33	0.2805	0.2772	0.5490	.....
4	0.53	0.2815	0.2762	0.5030	.....
9	0.48	0.2745	0.2697	0.3005	.....
10	1.06	0.2810	0.2704	0.2940	.....
11	1.57	0.2840	0.2683	0.2860	.....
12	1.64	0.2920	0.2756	0.2820	.....
13	1.71	0.3110	0.2939	0.2875	.....
10	1.06	0.2810	0.2704	0.2940	.....
14	1.19	0.2895	0.2726	0.3310	.....
15	0.92	0.2877	0.2728	0.3361	.....
16	0.44	0.2815	0.2851	0.4060	.....
18	0.74	0.2877	0.2803	.....	0.0132
19	1.44	0.2815	0.2671	.....	0.0203
20	1.51	0.2845	0.2694	.....	0.0410
21	2.25	0.2785	0.2560	.....	0.0542

Variation of Limestone and Coke in the Charge.—In Group I (Table III) the results of experiments Nos. 3, 4, and 5 indicate a constant relation between the carbon charged and that entering the pig steel. The pounds of carbon required for reduction only per pound of pig steel reduced varied from 0.2781 to 0.2762 lb., averaging 0.2771 lb. A higher figure was obtained than from the results of Group II, Table IV, which was probably due to the charge being more basic in the former case.

Variation of Coke in the Charge.—In Group II (Table IV) the lime charged was constant, although, in calculating it to pounds of pig steel reduced, it appears to be variable because of variation in the amount of pig steel reduced. From this group, for a pig steel containing from 0.48 to 1.57 per cent. carbon, there appears to be a constant amount of carbon required for reduction only, any added above this amount serving to carburize the pig steel. This amount varies between 0.2683 and 0.2704 lb., averaging 0.2694 lb. That is, if 0.2694 lb. of carbon were charged per pound of pig steel reduced, theoretically the product would be free from carbon. If 0.2794 lb. of carbon were used in the charge the product would contain 1.00 per cent. carbon. Above 1.57 per cent. carbon in the pig steel the excess of carbon in the charge does not seem to carburize the pig steel at the same rate.

Variation of Limestone in the Charge.—From the results of Group III (Table V), in which lime in the charge was varied, other factors remaining constant, with the increase of lime in the charge there seemed to be

TABLE III.—*Variation of Limestone and Coke in the Charge*

Experiment No.	1	2	3	4	5	6	7
Charge:							
Hematite, lb	15.00	15.00	15.00	15.00	15.00	15.00	15.00
Coke, lb	2.25	2.25	2.75	2.75	2.75	2.93	2.93
Limestone, lb	8.06	8.87	9.93	7.37	8.06	4.50	4.50
Product, steel:							
C, per cent	0.08	0.16	0.29	0.53	0.33	1.16	1.04
Mn, per cent	0.17	0.08	0.18	0.15	0.10	0.10	0.07
Si, per cent	0.07	0.04	0.02	0.07	0.04	0.06	0.08
P, per cent	0.088	0.067	0.057	0.063	0.082	0.030	0.010
S, per cent	0.027	0.045	0.050	0.045	0.020	0.051	0.033
Product, slag:							
SiO_2, per cent	28.40	22.88	22.32	22.20	26.36	24.86	33.28
CaO, per cent	37.46	40.25	46.20	41.76	45.60	45.40	35.80
MgO, per cent	11.60	12.61	5.61	6.60	7.25	7.65	7.98
FeO, per cent	14.23	12.85	6.33	8.58	7.59	7.85	7.87
Al_2O_3, per cent	5.52	5.58	17.01	19.07	7.47	10.95	11.81
Lb. carbon per lb. iron	0.213	0.213	0.213	0.263	0.263	0.263	0.286
Pig steel, lb	7.25	7.31	8.00	7.93	8.00	8.06	8.06
Per cent. recovery	84.95	85.60	93.10	92.30	93.00	93.40	93.20
Per cent. iron slagged	15.25	14.49	7.08	7.71	6.88	6.75	6.82
Kilowatt-hours per long ton of pig steel	4,550	4,375	5,620	4,280	3,750	5,440	4,910

TABLE IV.—*Variation of Coke in the Charge*

Experiment No.	9	10	11	12	13
Charge:					
Hematite, lb	15.00	15.00	15.00	15.00	15.00
Coke, lb	2.75	2.87	2.93	3.12	3.25
Limestone, lb	4.50	4.50	4.50	4.50	4.50
Product, steel:					
C, per cent	0.48	1.06	1.57	1.64	1.71
Mn, per cent	0.15	0.07	0.13	0.12	0.13
Si, per cent	0.12	0.13	0.16	0.23	0.24
P, per cent	0.026	0.041	0.089	0.107	0.121
S, per cent	0.147	0.085	0.067	0.048	0.044
Product, slag:					
SiO_2, per cent	31.28	36.73	37.24	41.20	39.75
CaO_2, per cent	39.03	36.25	30.20	32.16	34.85
MgO, per cent	10.90	7.75	11.61	8.84	10.70
FeO, per cent	8.73	5.26	2.53	0.50	3.50
Al_2O_3, per cent	7.12	5.88	12.58	17.45	10.50
Lb. carbon per lb. iron	0.263	0.274	0.286	0.299	0.311
Pig steel, lb	8.18	8.31	8.56	8.68	8.56
Per cent. recovery	94.45	96.20	98.30	99.60	97.60
Per cent. iron slagged	5.64	3.60	1.80	0.49	2.38
Kilowatt-hours per long ton of pig steel	4,830	4,910	4,115	4,030	4,375

a gradual increase of carbon required for reduction, and a corresponding decrease of carbon in the pig steel. This is probably due to the formation of calcium carbide with the increased percentage of lime present.

TABLE V.—*Variation of Limestone in the Charge*

Experiment No.	14	15	16
Charge:			
Hematite, lb	15.00	15.00	15.00
Coke, lb	2.87	2.87	2.87
Limestone, lb	5.00	5.50	6.00
Product, steel:			
C, per cent	1.19	0.92	0.44
Mn, per cent	0.08	0.07	0.08
Si, per cent	0.11	0.11	0.14
P, per cent	0.039	0.072	0.057
S, per cent	0.120	0.107	0.063
Product, slag:			
SiO_2, per cent	34.09	32.72	29.48
CaO, per cent	37.40	41.90	46.70
MgO, per cent	9.90	10.50	2.22
FeO, per cent	7.00	5.83	7.29
Al_2O_3, per cent	9.46	7.31	12.17
Lb. carbon per lb. iron	0.274	0.274	0.274
Pig steel, lb	8.25	8.31	8.12
Per cent. recovery	95.20	96.00	94.10
Per cent. iron slagged	4.88	3.97	4.92
Kilowatt-hours per long ton pig steel	6,000	4,375	5,900

In Group I, experiments 3, 4 and 5, with increased basicity of the charge the carbon in the pig steel decreases, while that used for reduction increases as in Group III. The decrease does not appear so marked with the more basic charge used in Group I. The carbon content of the charge was considerably less than that of Group III.

Variation of Fluorspar in the Charge.—In Group IV (Table VI) the increase of fluorspar in the charge causes a marked increase of carbon in the pig steel and a corresponding decrease of carbon used for reduction, due probably to excessive thinning of the slag.

The experiments of Neveu and Arnou[7] in 1910 in an intermittent electric furnace of the hearth type showed the ease with which pig steel could be made, using coke, anthracite, or charcoal as a reducing material. Hematite, magnetite, and siderite were reduced to pig steel, when charged in either a coarsely divided or a pulverulent state.

Trouble was experienced at the Hardanger, Norway, electric pig-iron plant with coke as a reducing material. The failure of this plant

[7] Arnou, M. G.: Notes on the Direct Reduction of Iron Ores in the Electric Furnace, *Revue de Métallurgie*, vol, vii, No. 12, p. 1190 (Dec., 1910).

TABLE VI.—*Variation of Fluorspar in the Charge*

Experiment No.	18	19	20	21
Charge:				
Hematite, lb	15.00	15.00	15.00	15.00
Coke, lb	2.87	2.87	2.87	2.87
Limestone, lb	5.87	5.81	5.56	5.43
Fluorspar, lb	0.12	0.19	0.37	0.51
Product, steel:				
C, per cent	0.74	1.44	1.51	2.25
Mn, per cent	0.11	0.10	0.08	0.10
Si, per cent	0.16	0.12	0.07	0.06
P, per cent	0.055	0.053	0.030	0.070
S, per cent	0.027	0.035	0.049	0.060
Product, slag:				
SiO_2, per cent	29.28	29.44	28.00	27.00
CaO, per cent	44.90	49.60	48.10	50.10
MgO, per cent	3.15	1.03	3.24	3.66
FeO, per cent	8.18	6.12	7.29	7.00
Al_2O_3, per cent	9.73	10.11	8.97	9.57
Lb. carbon per lb. iron	0.274	0.274	0.274	0.274
Pig steel, lb	8.12	8.31	8.25	8.37
Per cent. recovery	94.20	95.90	95.30	96.00
Per cent. iron slagged	5.73	4.08	4.70	4.36
Kilowatt-hours per long ton of pig steel.	9,900	5,280	6,880	.....

has been assigned to the use of coke, because coke has a higher electrical conductivity than charcoal and hence not as much heat is generated by the resistance of the charge to the passage of the electric current. For this reason, trouble might be experienced with a shaft furnace using coke as a reducing agent in the production of pig steel, but the problem will undoubtedly be solved eventually. The writer visited the Hardanger plant a few months before it was closed, and from observation of existing conditions does not believe the failure of the enterprise was entirely due to use of coke.

J. Crawford says[8] that at Heroult, while difficulty was experienced with coke alone as a reducing material, "by adopting certain precautions in crushing the stock and feeding the same into the furnace I have operated on a mixture of 60 per cent. coke and 40 per cent. charcoal with a very fair degree of furnace efficiency."

He further remarks: "I believe that many of our coals which make a poor metallurgical coke for blast furnace use on account of their low crushing strength might be found to make a satisfactory fuel for electric furnace use."

The experiments of Neveu and Arnou do not show anything regarding

[8] Crawford, J.: Progress of Electric Smelting at Heroult, California. *Metallurgical and Chemical Engineering*, vol. xi, No. 7, p. 383 (July, 1913).

the regulation of carbon in the product beyond that pig steels containing from 0.08 to 1.25 per cent. carbon were produced.

The results at both Domnarfvet and Trollhättan indicate that pig steel can be made directly from ore in the electric shaft furnace. In these runs no attempt was made to produce pig steel, but the product contained from 1 to 3 per cent. carbon, as before stated. While it would undoubtedly be more difficult to regulate the carbon of the product in a shaft furnace operated continuously than in an intermittent hearth furnace, it could be done, but the regulation would not be so close.

Hence it may be stated that:

1. In producing pig steel containing from 0.25 to 1.50 per cent. carbon for any particular furnace and basicity of charge, the carbon content of the pig steel can be regulated by varying the carbon in the charge.

2. An increase or decrease of lime in the charge causes a corresponding decrease or increase of carbon in the pig steel.

3. The use of fluorspar in excessive quantities causes a marked increase in the carburization of the pig steel.

4. The experiments of Neveu and Arnou indicate that the carbon of reduction may be supplied from coke, anthracite coal, or charcoal, and the charge may be fine, coarse, or briquetted.

5. The work at Domnarfvet and Trollhättan shows the possibility of keeping the carbon of the product below 2.2 per cent. in an electric shaft furnace.

Impurities in the Pig Steel

Silicon.—The results of Group II show that with an increased amount of coke in the charge there is a gradual, although not excessive, increase of silicon passing into the pig steel. The increased basicity of the charge, Group III, lowers the percentage of silicon in the pig steel. Increased charging of fluorspar seems to cause a decrease of silicon in the pig steel, Group IV. In general, the experiments show that the percentage of silicon in the pig steel can be kept at a low figure if desired, and that its regulation does not seem difficult.

This conclusion is corroborated by earlier investigators. In his first experiments, with an ore containing 2.79 per cent. SiO_2 and very pure charcoal as a reducing agent, Stassano obtained products with from a trace to 0.048 per cent. silicon. His second experiments, with ore containing 17.15 per cent. SiO_2, resulted in pig steel with from 0.03 to 0.22 per cent. silicon. Brown and Lathe,[9] using hematite with 2.23 per cent. SiO_2, had from a trace to 0.24 per cent. silicon in the pig steels produced. The work of Neveu and Arnou on hematite, magnetite, and siderite ores

[9] Stansfield, A.: *The Electric Furnace*, p. 133 (1907).

of from 0.6 to 8.96 per cent. SiO_2 gave products containing 0.02 to 0.19 per cent. silicon. Considering that the ore used contained 9.25 per cent. SiO_2 and the coke 13.62 per cent. SiO_2, the results of the writer, an average of 0.15 per cent. silicon in the product, show that no difficulty would be experienced with poor coke and impure ores.

Phosphorus.—Although the ore contained a high percentage of phosphorus, 0.124 per cent., no difficulty was experienced in keeping the percentage in the metal low. In the 20 experiments from 51.7 to 95.92 per cent. of the total phosphorus charged was slagged, with an average of 77.25 per cent. (See Table VII.)

In Group II, the gradual increase of coke in the charge caused an increase of phosphorus in the pig steel, because of the failure of the phosphorus to pass into the slag as calcium phosphate under the more reducing conditions of the operation. Increased lime in the charge, Group III, seems to result in an increased percentage of phosphorus in the pig steel. This was probably caused by the more basic slag being too thick for intimate contact with the steel bath during the agitation caused by the passage of the electric current through it.

Fluorspar, Group IV, causes a slight decrease of the percentage of phosphorus in the steel when used in small quantity, but when charged in excess it caused the percentage of phosphorus in the pig steel to increase. The slight thinning of the slag seemed to increase the slagging of the phosphorus. Continuous operation of the furnace did not cause increased phosphorus in the pig steel, as 89.3 per cent. of the phosphorus charged was slagged, and the pig steel produced contained 0.031 per cent. phosphorus.

These results agree with those of other investigators. Stassano in his first experiments used an ore containing 0.058 per cent. phosphorus, which resulted in pig steels with 0.005 to 0.024 per cent. In his second series the ore contained 0.15 per cent. phosphorus and the pig steel 0.01 to 0.10 per cent. Brown and Lathe, with an ore containing 1 per cent. phosphorus, obtained products with 0.031 to 0.49 per cent. phosphorus, but did not reduce the phosphorus in the pig steel to below 0.20 per cent. without excessive loss of iron in the slag. As the carbon content of the charge was decreased the percentage of phosphorus in the pig steel decreased, but with 0.091 per cent. carbon pig steel the FeO in the slag was 26.94 per cent. and phosphorus in the pig steel 0.031 per cent., whereas with 0.54 per cent. carbon the pig steel contained 0.20 per cent. phosphorus and the slag 3.6 per cent. FeO. In the work of Neveu and Arnou, with ores of 0.02 to 0.052 per cent. phosphorus content and pig-steel products of from 0.08 to 1.25 per cent. carbon, the phosphorus in the metal was reported as a trace.

Yngstrom asserts in his report on the Domnarfvet experiments that the phosphorus goes almost entirely to the iron. Considering the con-

TABLE VII.—*Elimination of Phosphorus and Sulphur*

Experiment No.	Phosphorus in Charge Grams	Phosphorus in Pig Steel Grams	Per Cent. of Phosphorus Slagged	Sulphur in Charge Grams	Sulphur in Pig Steel Grams	Per Cent. of Sulphur Slagged
1	9.45	2.91	69.20	14.98	0.89	94.05
2	9.48	2.23	76.40	14.98	1.50	89.98
3	9.71	2.07	78.75	16.32	1.82	88.82
4	9.61	2.28	76.40	16.32	1.72	89.42
5	9.55	2.98	68.85	16.32	0.73	95.53
6	9.71	1.10	88.70	16.91	1.88	88.85
8	9.60	0.37	95.92	16.91	1.21	92.85
9	9.52	0.96	89.96	16.32	5.45	66.60
10	9.57	1.16	88.00	16.66	3.21	80.72
11	9.60	3.46	63.90	16.91	2.61	84.57
12	9.66	4.25	56.05	17.26	1.90	89.00
13	9.70	4.70	51.70	17.58	1.67	90.49
14	9.54	1.46	84.70	16.91	4.49	73.40
15	9.55	2.72	71.55	16.91	4.04	76.20
16	9.58	2.16	77.40	16.91	2.31	86.35
18	9.57	2.03	78.80	16.91	1.00	94.18
19	9.57	2.00	79.15	16.91	1.32	92.20
20	9.56	1.12	88.29	16.91	1.83	89.17
21	9.56	2.67	72.15	16.91	2.29	86.42
23	19.29	2.06	89.30	33.14	3.22	96.80
Average			77.25			87.28

ditions of the runs this is to be expected, because, with the carbon necessary to produce a pig iron, the atmosphere of the furnace was too reducing to allow oxidation of the phosphorus. Also, the slags used in some runs were very acid, as high as 41.1 per cent. SiO_2. Leffler and Nyström in their 1912 report on the Trollhättan furnace[10] show results somewhat similar to those at Domnarfvet. From ores containing 0.002 to 0.055 per cent. phosphorus, metal was produced which contained 2.68 to 3.85 per cent. carbon and 0.013 to 0.045 per cent. phosphorus, with acid slags of from 36.54 to 45.15 per cent. SiO_2.

The conclusions drawn from these results are:

1. Increased carbon in the charge causes an increase of the percentage of phosphorus in the pig steel, because of the stronger reducing conditions.

2. Increased basicity of the slag does not cause the slagging of more phosphorus if the slag is very thick.

3. If a small amount of fluorspar is added to thin the basic slag, the slagging of the phosphorus is assisted.

4. Continuous operation of the furnace does not increase phosphorus in the pig steel.

[10] Leffler, J. A., and Nyström, E.: Electric-furnace Pig Iron at Trollhättan. *Metallurgical and Chemical Engineering*, vol. x, No. 7, p. 413 (July, 1912).

5. Owing to the more oxidizing conditions in the electric furnace when pig steel is being made rather than pig iron, a greater percentage of the phosphorus is slagged.

Sulphur.—The hematite used in the experiments contained 0.14 per cent. sulphur and the coke 0.54 per cent. Contrary to expectations, no difficulty was experienced in concentrating the sulphur in the slag. From 66.6 to 96.8 per cent. of the total sulphur in the charge was slagged, with an average of 87.28 per cent.

In Group II, the sulphur acted contrary to the phosphorus with the increased reducing atmosphere, being easily slagged, owing to the better conditions for the formation of calcium sulphide. In Group III, the increase of lime in the charge caused a marked increase in the amount of sulphur passing into the slag. The coke in the charge was low in amount, so that some of the sulphur probably passed into the slag as calcium sulphate. From Group IV, it seems that a small amount of fluorspar in the charge aids in the elimination of sulphur from the pig steel, but an excess causes it to pass into the pig steel. As in the case of phosphorus, more sulphur passed into the slag when the latter was fluid. Continuous operation of the furnace did not cause increased sulphur in the pig steel, as 96.8 per cent. of the sulphur charged was slagged, and the pig steel produced contained 0.048 per cent. sulphur.

In Stassano's first experiments the difficulty of elimination of sulphur was shown, as, from ore with 0.058 per cent. sulphur, the pig steels produced contained 0.046 to 0.073 per cent. sulphur. From the writer's experiments it appears that the slag must be very fluid and basic if sulphur is to be slagged in the making of pig steel, because of the weaker reducing atmosphere of the furnace. The slags used by Stassano were probably not basic enough. In his second series, from ore with 0.12 per cent. sulphur he obtained pig steels containing from 0.04 to 0.07 per cent. sulphur. Brown and Lathe, operating a furnace continuously, had difficulty in slagging sulphur. From ore containing 1 per cent. sulphur pig steel was produced which contained from 0.54 to 1.04 per cent. sulphur. This was probably due to either the furnace not being hot enough, or the slag, although basic, not being fluid enough. In some previous experiments by the writer these factors were found to be the cause of high content of phosphorus and sulphur in the pig steels produced. An illustration of the power of the electric furnace to slag sulphur with a heavy lime slag, if the slag is kept fluid by use of fluorspar, is shown in an experiment made by the writer for the production of ferro-molybdenum from molybdenite concentrate. There was 30 per cent. sulphur in the ore, while the ferro-molybdenum contained 0.19 per cent. sulphur, 0.73 per cent. carbon, and 50.55 per cent. molybdenum. Neveu and Arnou produced pig steel with 0.012 to 0.16 per cent. sulphur from ore containing 0.023 to 0.40 per cent. sulphur. The Domnarfvet

runs show that from a charge containing 0.50 per cent. sulphur a pig iron and occasionally a high-carbon pig steel was made containing less than 0.005 per cent. sulphur. Similar results were obtained at Trollhättan.

The conclusions formed are:

1. An increased reducing atmosphere aids in the passage of sulphur into the slag.

2. Increased basicity of the slag causes more of the sulphur to be slagged.

3. By thinning the slag a small amount of fluorspar assists in sulphur elimination from the pig steel, but an excess has the contrary effect.

4. Continuous operation of the furnace does not cause increased sulphur in the pig steel.

5. With a fluid basic slag, sulphur can be slagged readily in the electric furnace production of pig steel in both a continuous and an intermittent operation.

Loss of Iron in the Slag

The ferrous oxide in the slag was affected by the degree to which the operating conditions were reducing. With increase of coke in the charge the percentage of ferrous oxide in the slag was decreased, but was little affected by the variation of basicity of slag. The extremely high results in experiments Nos. 1 and 2 are due to the carbon in the charge being made slightly less in amount than the calculated, because a very low carbon pig steel was desired. The high loss of iron in the slag of the continuous run was caused by tappings of slag when unreduced ore was low in the furnace. Some of the unreduced ore passed into the slag. In the operation of such a small furnace continuously this would be a natural result, but it should not occur with a large furnace. The slags of all the experiments contained an average of 7.47 per cent. ferrous oxide. The average loss of iron in the slags was 6.09 per cent.

The results of the writer check closely the results of Neveu and Arnou, in which for a normal run the slag contained less than 8 per cent. ferrous oxide. Brown and Lathe had 3.6 per cent. FeO in the slag tapped with a 0.54 per cent. carbon pig steel, but had 26.94 per cent. ferrous oxide with a 0.091 per cent. carbon pig steel. In the writer's experiments, with a 0.08 per cent. carbon pig steel, there was 14.23 per cent. ferrous oxide in the slag. Evidently for pig steels containing less than 0.25 per cent. carbon there is apt to be high iron loss in the slag. At Domnarfvet, when a pig steel containing 1.70 per cent. carbon was to be tapped the slag contained 0.23 per cent. FeO, which checks closely with experiment No. 12, 1.64 per cent. carbon in the steel and 0.50 per cent. ferrous oxide in the slag.

With steady operation in a large furnace, the loss of iron in the slag

should not exceed 4 per cent. of the total iron charged, depending on the amount of carbon desired in the product, but if a pig steel containing less than 0.25 per cent. carbon is produced there will probably be a greater loss of iron in the slag.

Adaptability of the Process to Continuous Operation

To produce a pig steel directly from ore in the electric furnace at a cost low enough to compete even in remote regions with imported standard steels, a process must have possibilities of large output. While in the present development the electric pig-iron furnace is small in unit compared with the blast furnace, it will undoubtedly be eventually increased much above the present capacity. Applied to the production of pig steel, the electric pig-iron furnace offers much greater possibilities for large-scale work than any other furnace yet used in direct processes. In a shaft furnace of this type the energy consumption can be kept at a lower figure than in an intermittent furnace, because of the use of gases for preheating and reduction of the charge.

An experiment was made, extending over several hours, to note the effect upon the pig steel of continuous charging and tapping rather than intermittent operation of the furnace. In addition to hematite, limestone, and coke a small quantity of fluorspar was charged. A tapping was made about the middle of the run, which resulted in the following products:

Pig Steel	Per cent.	Slag	Per cent.
C	0.39	SiO_2	30.48
Mn	0.08	CaO	31.30
Si	0.12	MgO	13.39
P	0.031	FeO	15.20
S	0.048	Al_2O_3	9.28

The power consumption was 3,940 kw-hr. per long ton of pig steel. From the product it may be seen that as good a grade of pig steel can be produced in an electric furnace when operated continuously as when operated intermittently.

The results of electric-furnace pig-iron manufacture in Sweden also show the feasibility of pig-steel production in the electric shaft furnace, as regards carbon and impurities in the pig steel and loss of iron in the slag.

J. Crawford remarks as a result of his experience at Heroult with an electric shaft furnace: "The matter of too little carbon gives less trouble, and if the furnace is producing low silicon and carbon iron should give none at all." From his results it appears that it is easier to operate the shaft furnace when making pig steel than when making pig iron.

From the results available it seems that there should be no greater difficulty in the elimination of impurities from pig steel in a continuous shaft furnace than in intermittent operation.

The following process might be used for the direct production of pig steel from ore in a continuous electric shaft furnace: The mixture of ore, flux, and reducing agent is charged continuously into an electric shaft furnace so that the shaft is full all the time. A pig steel containing less than 2.2 per cent. carbon is tapped from the furnace into a ladle for transference to an open-hearth or electric furnace if further refining is necessary, or if not, into an externally heated mixer for deoxidizing and addition of ferro-alloys. While the carbon can be regulated to a certain extent by the quantity charged, if the production of a low-carbon steel, below 0.25 per cent. carbon, is attempted by reducing the amount of carbon in the charge, there will probably be excessive loss of iron by slagging. To avoid this loss advantage can be taken of the fact that the greater part of the reduction in the electric shaft furnace is performed by solid carbon in the upper part of the crucible. In the continuous experiment performed, it was noticed that there was a tendency for fused unreduced iron ore to work down into the bottom of the crucible. In spite of the presence of carbon in the charge sufficient to reduce the ore, when metal was tapped with the furnace full of charge the pig steel contained less carbon than the product from intermittent runs in which about the same amount of carbon was charged. For example, in experiment No. 4, 0.263 lb. of carbon was charged per pound of iron, and the pig steel contained 0.53 per cent. carbon, in comparison with 0.39 per cent. carbon in the pig steel tapped during a continuous run with 0.268 lb. of carbon per pound of iron. Experiment No. 9 shows a similar result.

This possibly may be explained by the fact that since there is only enough carbon for reduction present in the electric furnace, after the ore is reduced in the upper part of the crucible and the lower end of the shaft, there is not so great an opportunity for the iron to come in contact with the coke as is the case in blast-furnace smelting when sponge iron is carburized at the tuyères, so that a comparatively pure product settles out in the crucible of the electric furnace. Or, it is possible that, due to the fact that in the electric furnace reduction takes place in close proximity to the pig-steel bath, some unreduced ore works down into the bath and decarburizes it. This was apparently the case in some experiments on the production of ferro-chrome, when the percentage of carbon was considerably lower in the alloys produced by operation with the furnace shaft full at all times than with intermittent runs. In this case reduction could not have taken place very high up in the furnace, as chromite is not reduced until a temperature of 1,185° C. is reached.

A question might arise as to ease of tapping pig steel from a shaft furnace, because pig steel has a higher melting point than pig iron. The

writer had no greater difficulty in this respect with the small furnace used than is ordinarily experienced in experimental furnaces. In fact, 60 per cent. ferro-tungsten was readily tapped. In the experimental runs at Domnarfvet 1 per cent. carbon pig steels were tapped. It is probable that pig steel could be tapped from the Swedish type of electric shaft furnace as readily as pig iron, because of the concentration of heating in the crucible.

Cost of Production

Cost of production of pig steel from ore in the electric furnace is, of course, the main stumbling block to the success of such a process. As yet not enough practical work has been done to give conclusive figures. All work to date has been entirely experimental, and while it has shown that the metallurgical difficulties are not impossible to overcome, few data as to actual cost are available.

It may be stated, however, that in any place where there is a market for steel, and pig iron can be made at a profit in the electric furnace, it will be more profitable in the end to make pig steel for subsequent refining in an electric furnace or open hearth. than to produce electric-furnace pig iron with a subsequent refining to steel. This is true for two reasons:

1. The power consumption has been found to be lower per ton of pig steel than per ton of pig iron, which results also in a decreased labor cost per ton, as no more labor or capital outlay is necessary to operate the plant for pig steel than for pig iron.

2. As the pig steel contains a comparatively low percentage of carbon and impurities, the time necessary for refining is reduced, which results in a greater output at a lower cost.

The results of the early experiments at Domnarfvet show that the power consumption per ton of metal produced decreased as the amount of carbon charged per ton of iron was reduced.

In the latest report of the engineers at Trollhättan, it is stated that the power consumption per ton of pig iron varies in proportion to the iron content of the ore. A poor ore and pig iron high in silicon and manganese require more power than a rich ore and pig iron low in silicon and manganese. This was also found to be the case in California. The results of the writer, of course, show a high power consumption because of the small size of the furnace.

Among other expenses of pig-steel production, maintenance and capital charges would be about the same as for electric-furnace pig-iron production. The labor cost per ton of product would be a little lower for pig steel. The electrode consumption per long ton of pig iron has been reduced to 6.7 lb. It might be a little higher in making pig steel, because

of there being less carbon in the charge, but it should not exceed 15 lb. per ton of pig steel.

The use of pig steel in the open-hearth was tried out at Degerfors, Sweden. It was found that pig steel produced by the electric shaft furnace was more suitable for steel making in the open hearth than ordinary pig iron, and required less time for complete refining. Normal pig iron made in the electric furnace was found to be less suited to the production of open-hearth steel than normal blast-furnace pig iron. This shows the advantage of production of pig steel rather than pig iron in the electric furnace when steel is to be the final product.

Summary

1. In the electric-furnace production of pig steel from ore, carbon in the product can be kept below 2.2 per cent., and regulated to an extent by the amount of carbon charged, without resulting in excessive loss of iron in the slag or in the production of a pig steel very high in impurities, if a fair grade of ore is used.

2. It is not difficult to slag the greater part of the silicon, phosphorus, and sulphur of the charge, if the furnace is hot and the slag fluid, but conditions are less favorable to the slagging of sulphur than of other impurities in the operation of an electric furnace for pig-steel production, which is, of course, contrary to experience in the manufacture of pig iron.

3. The loss of iron in the slag should not be excessive unless the pig steel produced is of very low carbon content.

4. From the results with the Domnarfvet, Trollhättan, and Heroult furnaces, there does not appear to be great difficulty attending the production of pig steel in an electric shaft furnace, and in fact experience has shown that there is less difficulty in the operation of the electric furnace on pig steel than on pig iron.

5. At any place where there is a market demand for steel, and pig iron can be made in the electric furnace at a profit, the steel ultimately produced would be cheaper, if made by the electric reduction of iron ore to pig steel, followed by refining in another furnace if necessary, than if the product of the electric-reduction furnace was pig iron to be subsequently converted to steel in another furnace.

Discussion

J. W. Richards, So. Bethlehem, Pa.—I am extremely interested in this paper of Mr. Keeney's, which is one of the most elaborate published, so far, on the attempt to produce a low-carbon product directly in the electric pig-iron furnace. I think that the commercial side of it is very important. If we can make a low-carbon product, equivalent in composition, as far as carbon is concerned, to steel, and containing only moderate quantities of the other impurities, silicon, manganese, sulphur, and phosphorus, the refining of that pig steel to finished steel in the open-hearth furnace is a much easier operation than the refining of ordinary pig iron. That has been proved already. Several years ago I discussed that matter in connection with the Swedish electric pig-iron furnace, and I have always thought that it would eventually develop into commercial practice.

John Crawford, Jr., Heroult, Cal. (communication to the Secretary*).—The numerous and careful experiments carried out by Mr. Keeney on the possibilities of successfully making "pig steel" in the electric furnace have been studied by the writer with a great deal of interest. Judging from Mr. Keeney's conclusions, the five main advantages claimed for pig steel over normal low-silicon electric-furnace pig iron are:

1. That it can be converted into steel with greater facility and at a lower cost.
2. That it admits of greater economy in power consumption.
3. That it admits of greater economy in reducing material.
4. That the carbon content can be controlled within a limit of 2.20 per cent. maximum.
5. That the losses of iron in the slag will not exceed reasonable limits, say 6 per cent. FeO.

As to the first claim: From published reports, white electric-furnace pig iron, sufficiently low in silicon and carbon to approach or actually come within the limits of pig steel, has not only been used successfully but has met with a very favorable reception by Swedish steel makers. Hence a recital of our efforts to dispose of a similar product to steel makers on the Pacific Coast may be of interest. In May, 1912, there was in our stock yard about 150 tons of white iron which had been made some time previous in our shaft furnace. This was semi-porous and quite similar both in appearance and analysis to the pig iron which was at that time reported as being used successfully for steel making in Sweden. The analyses and weights of the several casts were as follows:

* Received Apr. 9, 1914.

Silicon Per Cent.	Carbon Per Cent.	Weight Tons
0.56	2.47	6
0.19	2.60	12
0.26	2.82	5
0.43	2.92	5
0.86	2.66	5
0.46	2.72	6
1.09	2.86	10
1.11	2.58	10
0.94	2.89	12
1.53	2.82	10
0.97	2.71	8
0.75	2.60	12
0.73	2.60	14
1.00	2.73	9
0.60	2.43	10
0.37	2.39	10
Average 0.74	Average 2.67	Total 144

The sulphur was uniformly under 0.03 per cent.; the phosphorus under 0.04 per cent.; and the manganese varied from 0.22 to 0.28 per cent.

Stimulated by the results which were being attained in Sweden we made considerable effort to induce steel makers on the Pacific Coast to try it. Finally by making a very attractive price we induced one firm to use it. This concern operates both basic and acid open hearths for making concrete reinforcing steel. The report of the works superintendent on it was rather indefinite but not enthusiastic. He said "It worked pretty well but normal pig iron was more satisfactory." Since that time we have sold this concern about 1,000 tons of normal low-silicon pig iron and also a few odd carloads of white iron which approached in analysis pig steel. They advise that no difference is noted in the working of normal electric-furnace pig iron as compared to normal blast-furnace iron of similar analysis. The pig iron brought the market price, but the pig steel always had to be offered on more favorable terms, and even at a reduced price it did not call forth any repeat orders.

These remarks are not intended to be construed as a contradiction of the favorable reception which pig steel has been accorded by the Swedish steel men, but it does seem to indicate that to gain the advantages claimed pig steel must be used "according to prescription" and a good deal of "missionary work" might be necessary before American steel makers, who have no patriotic interest in the matter, would do the necessary experimenting to prove the advantages claimed.

In regard to claims 2 and 3, while there is certainly an appreciable saving both in reducing material and in power in making low-silicon as against high-silicon iron in the electric furnace, how much this saving can be further increased by making pig steel instead of normal low-silicon pig iron is an open question. So far as the writer knows, no data have been published covering a continuous, systematic effort to make pig steel on a commercial scale. Messrs. Nyström and Leffler, in closing their report of a 215-day run in electric smelting at Trollhättan, make the statement: "The possibility of a very low carbon product—pig steel—has not yet been tried." As this report was made almost two years ago and no further data on making pig steel have been given out, it is fair to assume that either the possibilities of making it did not seem feasible or that if tried the results were not altogether successful. This opinion seems to be further verified by the statement of Ivar Barthen, Assistant Chief Engineer, Jernkontoret, Stockholm,[11] who says, referring to electric-furnace operations in Sweden: "The first expectations of the inventors may have been that only as much coal should be put in the [electric] blast-furnace as was needed by the chemical constitution of pig iron, as regards carbon; the heat indispensable to the process being secured by electric current. Even if those expectations could not be realized, still about 65 per cent. of the coal ordinarily consumed is saved."

We find in Messrs. Nyström and Leffler's report that for a period of 215 days' continuous operation, during which over 3,000 tons of iron was made, the silicon averaged only 0.72 per cent., which seems to indicate that although they were not endeavoring to make pig steel the metal produced was approaching it more nearly than it was normal pig iron. Since there are no data available to indicate that the fuel consumption has been lowered, we assume that the figures as given in their report are still substantially correct. It may be interesting, therefore, to make some comparisons between carbon economy attained in making low-silicon pig iron in Sweden, the results obtained by Mr. Keeney in his experimental furnace, and those obtained in producing the few casts of low-silicon iron previously noted as having been made at Heroult in a shaft-type furnace.

From the Trollhättan report we gain the following information:

Pig iron produced (95.36 per cent. Fe), metric tons	3,215
Coal charged (72.89 per cent. C), metric tons	1,301
Per cent. carbon in pig iron (average)	3.406
Per cent. silicon in pig iron (average)	0.72
Per cent. FeO in slag (mean)	3.40

[11] *Iron Age*, vol. xciii, No. 4, p. 253 (Jan. 22, 1914).

Furnace	Per Cent. Carbon Used to Iron Produced	Per Cent. Carbon Used in Reduction and Losses to Iron Produced	Carbon in Iron, Per Cent.	Silicon in Iron, Per Cent.	FeO in Slag, Per Cent.
Trollhättan.....	29.5	26.1	3.40	0.72	3.40
Keeney.....	28.7	27.6	1.07	0.11	6.56
Heroult.....	29.9	27.2	2.67	0.74	4.30

A comparison of these figures shows that though Mr. Keeney has made pig steel very low in carbon and silicon, the per cent. of total carbon consumed is little better, considering the greater loss of iron in the slag, than is attained in shaft-furnace practice when making low-silicon iron. The reason for this lies in the fact that in the shaft furnace the carbon used in reduction is oxidized to CO and CO_2 in the ratio of about 2.5 to 3 CO to 1 CO_2 by volume (the average given by Nyström and Leffler is 2.68 to 1), whereas in the small rectangular furnace used by Mr. Keeney we find the average ratio CO to CO_2 to be 4.7 to 1, with a minimum of 1 to 1 in experiments Nos. 1 and 2, where the FeO loss in the slag was 14 and 13 per cent. respectively, and a maximum of 11.2 to 1 in experiment No. 12, where the FeO loss in the slag was only 0.50 per cent. (Experiment No. 13 was excluded as it appears erratic.)

If we recast Mr. Keeney's figures on a basis of $CO : CO_2 :: 2.75 : 1$, which makes them comparable[12] with results attained in the shaft-type furnace, we have:

Furnace	Per Cent. Carbon Used to Iron Produced	Per Cent. Carbon Used in Reduction and Losses to Iron Produced	Carbon in Iron, Per Cent.	Silicon in Iron, Per Cent.	Fe in Slag, Per Cent.
Trollhättan, shaft type.	29.5	26.1	3.40	0.72	3.40
Keeney, shaft type	25.5	24.4	1.07	0.11	6.56
Heroult, shaft type.	29.9	27.2	2.67	0.74	4.30

These revised Keeney figures show a saving in carbon of 4 to 4.5 per cent., or an economy in reducing material of about 5 to 6 per cent. The

[12] Since at Heroult the circulating gases were not scrubbed and since at Trollhättan the moisture was reduced to 0.5 g. per cubic meter, and because part of the carbon consumed in reducing CO_2 in the circulating gases to CO is later recovered by the reducing action of the CO on the ore, no account was taken of these in revising Mr. Keeney's figures. Though not strictly accurate, the figures given are close enough for comparison.

metal is low in carbon and silicon and the slag loss is not unreasonable. They are certainly very attractive, but can they be attained or even closely approximated in actual practice? The writer does not think so. He believes we have already attained very close to the practical limit of carbon economy in the types of furnaces existent. Any further appreciable decrease in reducing material will be at the expense of lower recovery.

Mr. Keeney cites (p. 175, Loss of Iron in the Slag), a result obtained at Domnarfvet, where pig steel running 1.70 per cent. C was accompanied by a slag containing only 0.23 per cent. FeO. In the absence of knowledge of the fuel ratio used and the analyses of the slag and metal taps immediately preceding and following it is hardly safe to make any observations on this, but in experiment No. 12, which Mr. Keeney cites as analogous, we note the carbon used is 29.8 per cent. of pig iron produced, which cannot be considered any economy over present shaft-furnace practice.

The writer's justification for the opinions given in the preceding paragraphs is based on observations of both shaft-type and rectangular-type furnace at Heroult and on such data as have been available concerning electric iron furnaces, operating elsewhere. The results of his deductions may be summarized as follows:

The theoretical amount of carbon required to make pig metal from iron ore depends:

1. On the ratio in which the carbon combines with the oxygen in the ore to form CO and CO_2.
2. On the carbon absorbed by the metal.
3. On the carbon used in reducing silica and the other metalloids.
4. On the carbon consumed by oxygen dissociated from aqueous vapor and atmospheric oxygen occluded in the reducing material and sucked into the furnace through cracks.
5. On the portion of the carbon consumed in reducing CO_2 to CO which is not later recovered by the reducing action of the latter on the ore.

With sufficient carbon present to satisfy these requirements we should make pig iron, with an accompanying slag free from iron. But whether we do or not depends on another factor, which, though understood by all metallurgists, does not seem to have been given the importance it merits in electric smelting; this is, the ratio of size of the reducing material to the ore and the homogeneity of mixture of the two when they have descended to the smelting zone. Even in the shaft-type furnace, where a considerable reduction by CO can be counted on, much ore reaches the smelting zone of the furnace wholly or partly unreduced. Whether this is smelted at this point to an oxide which combines with the silica of the slag, or is reduced to metal, depends on the intimacy of its contact with carbon. Our experience with the shaft-type furnace indicated that even with a brittle reducing agent, like charcoal, if an excess over theoretical

calculations was not used the slag losses were erratic and generally high. The absorption of carbon by the metal was also erratic, but it did not always vary in inverse ratio to the slag losses, which might be expected and is indicated by Mr. Keeney's experiments. Obviously, if lump coke is substituted for charcoal conditions will be much worse. On one occasion it was necessary for the writer to be absent when one of our rectangular furnaces was being started upon coke, and the furnace man in charge failed to follow instructions *re* crushing the coke. On his return, two days later, the writer found melted oxides were being tapped from the furnace, and practically no metal. Without altering the amount, coke crushed to the proper size was used in the charge, and within a short time normal foundry iron was being produced.

Hence it is apparent that to keep slag losses within reasonable limits we must either use a considerable excess of lump coke or use crushed coke. The former method will result in an accumulation of coke in the crucible, low voltage, excessive current densities, high power consumption, poor power factor, melting roof, etc. The latter method, while presenting no metallurgical or electrical trouble, would in a shaft-type furnace make the burden so dense as to offer too much resistance to the ascending gases and would further aggravate the natural tendency of coke to "hang up," with subsequent annoying if not dangerous "slips." Crushed coke has been used satisfactorily by us in our rectangular furnaces for making all grades of foundry iron and doubtless could be utilized for making pig steel without excessive slag losses. But in furnaces of the rectangular type there is little reduction by CO in the stacks, hence the ratio of CO to CO_2 is high and the desired fuel economy is not attained.

To summarize, our experience at Heroult has indicated:

1. That though pig steel may have signal advantages over normal low-silicon pig iron for steel making, American steel makers will probably be found to be "from Missouri" and "will have to be shown."

2. That though there is considerable economy in both carbon and power in making low-silicon iron as against high-silicon iron, carbon requirements cannot be made practically to approximate the theoretically calculated amount.

3. That pig steel of 2.20 per cent. maximum carbon can be made in existing types of rectangular electric furnaces with reasonable slag losses, but with high carbon consumption as compared to shaft-type furnaces.

4. That pig steel of 2.20 per cent. maximum carbon cannot be made practically in existing types of electric shaft furnaces with carbon economy approaching theoretical figures except by accepting a high loss of iron in the slag.

5. That pig steel can be made with lower power consumption than normal low-silicon pig iron, but the saving per unit of production will be mitigated by the lower percentage of recovery.

6. That the best commercial possibilities for making pig steel with low fuel consumption and high percentage of recovery seem to be offered by the use of briquets of concentrates and reducing material, or flue dust and reducing material, in a furnace of the shaft type.

Though not properly a part of this discussion, since Mr. Keeney mentions "Trouble was experienced at the Hardanger, Norway, electric pig-iron plant with coke as a reducing material," it may be of interest to briefly summarize our deductions from 90 days of operating our rectangular furnaces on coke.

1. Any ordinary grade of coke, by crushing it, can be used satisfactorily from both a metallurgical and an electrical standpoint to produce any normal grade of pig iron. But, depending on its porosity and crushing strength and its analysis, there is for each variety a size, to be determined by experiment, which will give from all standpoints best operating conditions.

2. Soft, porous cokes are to be preferred over hard, dense cokes, because the former can be fed in larger pieces and still insure homogeneity of mixture when the burden descends to the smelting zone. Further, they cost less to crush, and there is less loss in fines and flue dust because of the larger size which can be used. For ore crushed to pass a $2\frac{3}{4}$-in. grizzly a soft, porous coke gives best results when crushed to pass a $1\frac{1}{2}$-in. grizzly. Certain hard, dense cokes had to be crushed to pass a $\frac{3}{4}$-in. grizzly.

3. If the coke is of the proper size when it reaches the smelting zone, operations may be carried on with as high voltage, low current densities, good power factor, and as cool a roof when using coke as when using charcoal.

4. The power consumption per unit of production is higher with coke as a reducing agent than with charcoal, but the increase is due to the extra amount of slag made. This depends on the composition and amount of ash in the coke and the amount of sulphur in the coke.

5. The carbon consumption per unit of production is higher with coke than with charcoal. This is due partly to the greater flue-dust losses from crushed coke than from lump charcoal and partly to the formation of carbides, in the high-lime, desulphurizing slag, all of which are not subsequently decomposed.

6. If electrodes are regulated to form a free-burning arc the use of coke has no bearing on the electrode consumption. If, however, electrodes are inserted in the charge to form a submerged arc, as is our practice, the consumption of electrode material is considerably greater, especially with graphite electrodes. This is due principally to the greater abrasive action of coke than charcoal, though partly also to the attack of the lime slags on the electrode material. We found the electrode loss from breakage (using 12-in. graphite electrodes) very excessive, doubtless

due in part to the greater density of the burden when using crushed coke as against lump charcoal. On this account, after carrying our experiments far enough to demonstrate the commercial feasibility of operating on coke, we have shut our furnaces down pending alterations to permit the use of 24-in. carbon electrodes, which, besides being stronger, will better resist the abrasive action of the burden.

7. With a coke running 1.25 per cent. sulphur we had little difficulty in keeping the sulphur in the pig below 0.05 per cent., and much of the iron made was under 0.03 per cent. sulphur.

8. No trouble was found in making foundry irons up to 3.50 per cent. silicon or higher if desired. In fact, on account of its high, easily reduced, siliceous ash, coke will produce a high-silicon iron somewhat more readily than charcoal.

9. Except for a somewhat more open grain, no difference has been noted thus far in the physical characteristics of iron made with coke and iron made with charcoal.

10. Crushed coke, especially when high in ash and with a large percentage of fines present, has a tendency to cause the burden to hang. This we believe would prohibit its use successfully in a furnace of the shaft type. However, in the rectangular furnaces there is an opportunity to dislodge scaffolds by barring down, if the hang is of any duration. There are several ways in which the presence of a hang up can be detected before it has become serious. A mixture of 15 to 25 per cent. of lump charcoal with the coke is quite effective in keeping the burden from hanging up.

In conclusion, we may say that coke has nothing to recommend it over charcoal for an electric-furnace reducing material excepting the difference in price. However, "black-butts," screenings, and even breeze if the dust is removed, can be mixed to make a satisfactory reducing material. As such grades of coke can usually be secured at a very attractive price this may more than offset the disadvantages of its use.

The writer apologizes for offering the above statements as "glittering generalities" unsubstantiated by definite figures, but, since the data obtained varied considerably with the chemical and physical properties of each coke tried and the size to which it was crushed, even a synopsis of this rather voluminous data is considered beyond the scope of this discussion.

Valuation of Iron Mines

Further discussion of the paper of JAMES R. FINLAY, *Trans.*, xlv, 282, with special reference to the discussion by R. B. BRINSMADE.

BY E. E. WHITE, ISHPEMING, MICH.

I DISAGREE with Mr. Brinsmade on several points, and I will discuss his remarks under the same six topics which he has used.

1. *Rate of Interest Earned by Sinking-Fund.*—I believe Mr. Brinsmade is confused in his ideas as to the effect of new gold on the interest earned on money. The flood of gold which he mentions certainly should not operate to increase interest rates, but since it replaces and adds to liquid capital should reduce interest rates and counteract the destruction of liquid capital which he mentions. Furthermore, it is doubtful if the destruction of capital since 1897 by wars and catastrophes has been any greater in proportion than in similar periods before 1897. With regard to savings banks, I would say that only 3 per cent. is allowed on deposits through a large part of the West. As to bonds, they can be bought to yield any rate of interest desired up to 6 per cent., but a 5 per cent. bond is not as safe as a 4 per cent. bond, nor a 4 per cent. bond as safe as a 3 per cent. government security.

I do not understand why Mr. Brinsmade says that Mr. Denny's quoted opinion of a 3 per cent. rate is *evidently* based on English conditions before the Boer war. Certainly there is no reference to the Boer war or to English conditions in Mr. Denny's article. M. Howard Burnham, in his admirable work on mine valuation, says:[1] "For instance, redemption is taken at 3 per cent., because even 4 per cent. represents a greater risk, as is indicated by the purchase of Government paper to bear only 3 per cent., the great desideratum of the latter being security." Again, Mr. Burnham says:[2] "The tables are calculated on the assumption that the dividends shall accumulate at 3 per cent. only, as in accordance with the general idea set out elsewhere, a rate greater than 3 per cent. involves appreciable risk with which it were inaccurate to saddle the sinking-fund if risks are to be segregated, this being the basis of all the writer's calculations."

Consols, the best-known government security in the world, are now (December, 1913) selling at about 72, yielding 3.48 per cent. on that price; French Rentes are selling to yield 3.5 per cent. and German 3 per cents to yield 3.95 per cent. Every one knows, however, that bonds and investment stocks all over the world are depressed at the present time, due to

[1] Burnham, M. Howard: *Modern Mine Valuation*, p. 19 (1912).
[2] *Idem*, p. 134.

straining of credit by the last few years of great prosperity, and the consequent scarcity of money. The usual return on Consols is lower than at present, and it has only averaged higher for one year in the last 75 years—in 1848, when it averaged 3.51 per cent. In 1897 it averaged 2.45 per cent., and in only five of the last 30 years has it averaged higher than 3 per cent. It therefore seems to me that Mr. Denny and Mr. Burnham are amply justified in assuming that 3 per cent. is the rate of interest that should be expected for a minimum risk.

2. *Rate of Interest Earned by Investment.*—I am not at all certain that Mr. Denny referred to gold mines with uncertain and scanty ore reserves in his statement of mining risks, as Mr. Brinsmade says. The Rand cannot be included in this class and I certainly believe it fair to assume that he included the Rand in his general statement of mining risk. A statement from Mr. Denny would be valuable in this connection, also from Mr. Hammond, whom I quoted in my discussion of Mr. Finlay's paper.

I disagree with Mr. Brinsmade in his comparison of iron mines and city real estate. As Mr. Brinsmade says, city real estate is speculative, but a decrease of 25 per cent. in the gross income only means a decrease of approximately 50 per cent. in the net income and therefore in the value, whereas in the case of most iron mines a 25 per cent. cut in ore prices would mean a decrease of 100 per cent. in the net income; that is, would wipe out all the profit and all value. For instance, assume taxes, repairs, maintenance, etc., to be 5 per cent. of the value of real estate, then the rent must be 10 per cent. to return 5 per cent. net on the investment. If the rent falls to $7\frac{1}{2}$ per cent. the net income would be $2\frac{1}{2}$ per cent. In the case of iron ore, however, the selling price is roughly $4 per ton, and the profit rarely over $1 per ton, usually much less. A decrease of 25 per cent. in the value per ton would therefore take all the profit. In other words, the cost of production is a so much larger proportion in the case of iron ore that the two cannot be fairly said to resemble each other. I believe the low rate of interest accepted by investors in real estate is partly due to the fact that they expect an unearned increment—*i.e.*, an increase in the value of their property due to the growth of the city. Investors in real estate are willing to pay more, and value the property upon a 5 per cent. basis, for the same reason. There is little chance of an increase in the value of an iron mine, where the ore is already blocked out. There is almost no chance of a permanent increase in the selling price of the ore, so that any increase in the value of such a mine must be due to a decrease in the cost of production. The tendency of the latter is in the wrong direction at present.

Mr. Finlay applied his method and the same rate of interest to all Michigan mines, regardless of whether or not there was any speculative value due to possible increase in the ore reserves. Where mines are valued

on the assumption that the ore body continues for 100 ft. below the bottom level, or some other conservative arbitrary distance, there is a possibility of increase in the ore reserves and in the value, as in the case of real estate. Where the limits of the ore bodies are completely known, however, either by underground work or by diamond drilling, or where any possible extension of an ore body is cut off by the property lines, this speculative value does not exist, and the return on the investment should be correspondingly greater. By using the same rate of interest in valuing all mines by this method, an injustice is done mines which are fully developed, and particularly certain mines where the ore bodies are in flat-lying beds, and where the whole tonnage may therefore be calculated from the results of drilling. J. Parke Channing discusses this point in his remarks on "Mine Valuation." He says:[3] "May it not be possible to have a mine with too much ore in sight? I ask this question because the general tenor of the discussion on mine valuation has made ore in sight the crucial test of the value of a property. If based on this test, many of our good iron mines would make a sorry showing, and yet their capacity for production is well known and year after year they come up to expectation." Speaking of certain pyrites mines, Mr. Channing goes on to say: "We have taken the ground that in mines of this character the deposits are so regular that the better test of their relative values is to be found in their capacity for production per 100 ft. of depth, rather than in the number of tons of ore available for immediate shipment; that is, in sight." In other words, Mr. Channing claims that mines should not necessarily be valued in proportion to their ore reserves, and that if they are, the mines with the most ore in sight are overvalued. This is the same as saying that if ore reserves are taken into account, as they must be, the valuations of mines with large ore reserves should be obtained by capitalizing the profits from the reserves at a higher rate of interest than in the case of steady producers with small ore reserves but considerable speculative value.

Another important difference between iron mines and city real estate is the amount of physical risk to the property. It must be admitted that the risk of fires, cave-ins, strikes, sand-runs, breakdowns of machinery, etc., in the case of mines, is greater than the risk of fires, floods, tornadoes, etc., in the case of city real estate. Furthermore, the cost of insurance against these risks in the case of real estate is comparatively small, and is included in the fixed charges. In the case of mines no insurance against the risks mentioned above is carried, as its cost would probably be prohibitive, and the damages difficult to ascertain.

Another condition that enhances the value of city real estate, and so affects the rate of interest, is the possibility of ready sale without material sacrifice. Iron mines are only sold at long intervals.

[3] *Engineering and Mining Journal*, vol. lxxvi, No. 11, p. 383 (Sept. 12, 1903).

The differences between iron mines and real estate are, briefly, the greater stability of profits in the case of real estate, the lesser physical risk and possibility of covering it with insurance, the greater chance of an increase in the principal (as compared with most iron mines), and the possibility of ready sale. In view of these differences it certainly seems reasonable that the return upon an investment in iron mines should be considerably greater than upon an investment in real estate.

As regards stock-market valuations, Mr. Finlay realized that mines are much overvalued in the stock market and disregarded these valuations in his valuation of Michigan copper mines. This overvaluation is probably because stocks are a medium of speculation and most investors in mining stocks expect to make more from an increase of principal than from dividends. Because speculators are satisfied with 5 per cent. dividends from listed mining stocks is no reason that that is a fair return on money. They all expect to make more than that, whether they do or not. I would ask Mr. Brinsmade if he would invest in mining stocks expecting to net only 5 per cent. after charging off part of his dividends for a sinking fund, or if he values mining properties for his clients on a 5 per cent. basis.

Mr. Burnham says:[4] "It is a source of constant wonder that people invest in copper or silver ventures on the same basis as gold—*i.e.*, that they accept, say, 6 per cent. return on copper shares, and ask the same of gold. If such investors could bring themselves to abandon their attitude of self-deception for a moment they would no doubt admit that what they really want is share speculation, but in order to indulge themselves comfortably in such pastime, they seek to be told that it is investment."

In this paragraph Mr. Burnham had reference to the variable selling price of other metals than gold, and iron ore falls in the same class as copper and silver. I believe that 5 per cent. is altogether too low a return to make an investment in iron mining attractive, and therefore too low a rate to use as a basis of valuation.

It has been claimed that in a valuation such as Mr. Finlay's all risks are taken into account in the average cost of production. This might be true of certain hazards of the mining business, such as strikes, mine fires, cave-ins, sand-runs, etc., if they have happened during the period for which costs are averaged. They probably have not happened in such a short period, however, and there is always the risk that they may be worse in the future than in the past. But, aside from these risks, there are certain risks that cannot possibly be taken account of in the cost of production. These are the risk that the future average price of ore will be less than the average price assumed, the risk that the estimate of tonnage, often based largely on diamond drilling, is too great, and the risk that for one reason or another the annual production, depending upon the amount that can be sold

[4] *Modern Mine Valuation*, p. 26.

annually, may not average as high in the future as in the past. These risks can only be taken into account by a rate of interest higher than the normal rate of 5 per cent., and necessitate a rate of at least 10 per cent.

3. *Place of Royalty Item in Valuation.*—From Mr. Brinsmade's discussion it is evident that he misread my paper. I had no idea of adding royalty to mining cost in reckoning valuation for taxation purposes and did not do so in my discussion of Mr. Finlay's figures. I considered royalty in comparing the cost of production of Lake Superior and Cuban ores, for reasons stated, but that is a very different thing.

If we were reckoning valuations from the standpoint of a purchaser it would nevertheless be entirely correct to add royalty to mining cost, and I therefore disagree with Mr. Brinsmade's last paragraph on this subject. Theoretically royalties may not affect the scale of prices, but in practice they do when an industry is firmly established on a royalty basis with long-term leases, as is the case in the Lake Superior district, or in the anthracite field in Pennsylvania. If conditions change so there is no profit to the operator on this royalty scale, I admit that the average royalty would tend to decrease, but it would be a slow process and would probably take at least 10 years to effect any considerable decrease in the royalty. A purchaser could not count on such a future reduction and must consider royalty as a mining cost. From the standpoint of the State, royalty is immaterial, however, since it makes no difference to the State whether the profit goes to the operator or to the fee owner in form of royalty. The valuation for taxation purposes is necessarily reached by including royalty as profit.

4. *Place of Taxation Item in Valuation.*—Mr. Brinsmade's suggestion is very good theoretically, but I do not believe it is practical. If the only taxes to be levied were State taxes, it might do, but with the complication of city, township, and county taxes brought in, different for almost every mine, it would be very difficult to apply. No values could be placed on the mines until the preliminary valuation of each city, township, and county and the total valuation of the State were determined and the amount of money to be raised for each. "t" would then have to be determined independently for each mine three times, for city, county, and State taxes, and the sum of the three t's used to figure the final valuation.

It seems to me simpler to apportion each mine its share of the taxes for the current year as nearly as can be estimated and divide by the average tonnage per year, adding the cost of taxes per ton to mining cost. The result should be the same as by Mr. Brinsmade's suggested method.

5. *Calculation of Future Ore Prices.*—Mr. Brinsmade treats Mr. Finlay's assumed future price of Lake Superior ore too lightly. To be sure, it is not so bad for a company to pay $100,000 more than its fair share of taxes for one year as it would be to pay the extra $100,000 for 10 years, but it is bad enough. Furthermore, it is just as bad to have the

erroneous assumption made every year for 10 years as it would be to have it made in the beginning to hold for 10 years. In Michigan, Mr. Finlay's valuations stood for two years, and when they were revised, I believe, the future price of ore was determined from the average of ore prices for the last five years instead of Mr. Finlay's period of seven years. If the revaluation were made every year and the same period used each year to determine the average price, I admit that the errors in determining the future price would balance in the course of time, but what security is there that the same period will be used each year, with changing tax commissions and different mining experts? After a period of low prices when mines have been valued high because of the high average price during the previous period it will be a great temptation to the tax commission, when prices rise again, to change the period used to determine the average price so as to obtain a higher average, more nearly corresponding with current prices. In this way the mines would always be taxed on the basis of high prices. In other words, the human and political element must be considered, and the future average price must be forecasted as accurately as possible, not depending on errors to balance.

6. *Valuation of Undeveloped Mining Land.*—"Self-assessment" may be practical for timber and real estate where the value is in sight and may be estimated closely. In the case of undeveloped mining property, however, the owner does not know the true value, and has no means of determining it except by expensive exploration. It surely is not right to force him to explore it, involving not only the expenditure of large sums of money but a large annual loss in interest on this expenditure until such time as the property can be developed. It would be suicidal for an operator to attempt to explore and develop all his undeveloped land at once, yet that is what "self-assessment" would lead to. Such a policy would result in over-expansion of the operator, overproduction of iron ore, and collapse in the first period of business depression. Would Mr. Brinsmade wish to force an overproduction of iron ore similar to the overproduction of bituminous steam coal, or rather the over-capacity of production, in this country? It is well known that most soft-coal mines are closed down a part of the year. Iron mining is now a prosperous industry and the miners are well housed and fed and have steady work all the year round as a rule. There is over-capacity enough in the iron business now without forcing development of all undeveloped lands at once. "Self-assessment" is diametrically opposed to conservation, to which more and more attention is being directed of late.

If "self-assessment" did not lead to over-development and overproduction, it would lead to confiscation of property by the State, and we have not yet reached that stage in this country. The State has no right to force an owner to sell or to name a price at which he will sell.

Another disadvantage would be the possibility of the land becoming

more valuable before a second assessment. It would be a temptation to graft and dishonesty on the part of office holders, who might confiscate the property and sell to more favored parties. Others would probably know of the increased value long before the owners, and by conniving with State officials might confiscate it.

Summary.—In Mr. Brinsmade's summary he states that Mr. Finlay's general scheme of mine valuation has been sustained by the Michigan courts. This is true, as far as it goes, but it must be borne in mind that it was only the general scheme that was ruled upon in the Newport and Sunday Lake cases, to which Mr. Brinsmade undoubtedly refers; that the factors used by Mr. Finlay were not specifically confirmed; and that these cases have only been decided in the lower court. The Newport case has been heard in the Supreme Court but has not yet been decided. The Sunday Lake case has been appealed, but has not yet been heard in the Supreme Court.

Mr. Brinsmade must be unfamiliar with the terms of leases in the Lake Superior district, since he commiserates "any operators who . . . were unwise enough to take long leases on a fixed royalty with the proviso that the lessee should pay all taxes." As a matter of fact, *all* operators have to take leases either on a flat rate royalty or on a percentage of the selling price per ton, since those are the only royalty terms ever given in the district. A percentage basis is worse for the operator than a flat rate, since as taxes increase the selling price increases and hence the royalty increases. A percentage of net profit would be more desirable from the operator's standpoint, but I know of no such leases in the district. It is doubtful if fee owners would ever agree to such an arrangement, since it would be apt to lead to falsification of records and wrongly kept cost sheets, and with mismanagement they would receive little or no royalty.

R. B. Brinsmade, Puebla, Pue., Mexico (communication to the Secretary*).—In answer to Mr. White's criticisms I beg leave to offer the following explanations, using the same six topics as before:

1 and 2. *Rates of Interest Earned by Sinking Fund and Investment.*—Mr. White is correct only as to the *ultimate* effect of a flood of gold, which, being liquid capital, would undoubtedly tend to lower the rate of interest. But the *transient* effect of a rapid and continuous increase in the money metal is the opposite, on account of its boosting effect on commodity prices. The rise in Sauerbeck's index of average prices, which has taken place since 1896, has forced creditors to protect themselves by a sufficient increase in the rate of interest to cover the expected loss in the purchasing power of their loaned money when the debt will be repaid. Thus, if the normal rate were 4 per cent., the interest would be advanced to 5

*Received Jan. 16, 1915.

per cent. whenever the purchasing power of money was falling 1 per cent. annually.

In discussing the fixation of rates of interest, we have the personal factor as the deciding one, when all is said. In the case of sinking funds, opinions will differ as to what class of bonds offer perfect security for capital, and in the case of mining investments two opinions will never completely agree as to what constitutes a fair return to cover the risk. I would not hesitate to recommend a 5 per cent. mining investment to a client who wished to take no chances; but as most investors in mines are playing for large profits they are generally prepared to undergo corresponding risks of loss. But though these variable and problematical rates of interest might possibly be used in the reckoning of valuation for taxation, as proposed by Mr. White, I believe there is less liability of error in Mr. Finlay's method, of adopting a constant rate of interest for all mines, and compensating for the speculative factors by varying the additions made for the value of probable and possible ore.

3. *Place of Royalty Item in Valuation.*—Royalties cannot affect the scale of prices, any more *practically* than *theoretically*, except where prices are no longer on a competitive basis, *i.e.*, fixed by the cost of reproduction on marginal land.[5] In this connection, Mr. White, in his comparison of the Lake Superior iron with the Pennsylvania anthracite district, seems to be inaccurate. At present, the anthracite field is controlled by a railroad combination[6] which fixes the price of coal to the consumer at "what the traffic will bear;" a monopoly price which has no necessary relation to the competitive factors which fix the price of Lake Superior iron ore. Therefore, though royalties may burden the operators, who have to pay them, they cannot affect the market price of iron ore as long as the United States consumer can range for his purchases not only over six Lake Superior districts and the South, but over Canada, Latin America, and Europe.

4. *Place of Taxation Item in Valuation.*—I accept Mr. White's opinion that my method of calculating the rate of taxation would be unsuitable in Michigan, because of the fiscal anarchy there, which permits direct taxation by four independent bodies. The Latin system, which confines real-estate taxation to one central office in each State, is certainly vastly superior to such frenzied localism.

5. *Calculation of Future Ore Prices.*—I admit that Mr. White has some grounds for his contention as to the liability to a cumulative error in the forecasting of future ore prices. But this uncertainty should not be

[5] R. B. Brinsmade: Discussion on Our National Resources and Our Federal Government, *Trans.*, xliv, 633 to 640 (1912).

[6] L. L. Redding: King Coal and His Court, *Everybody's Magazine* (New York, 1912).

blamed on Mr. Finlay's method of forecast, which seems as accurate a one as any other, when the true culprit is the method of assessment. In my new book[7] of suggestions for the economic reconstruction of Mexico (which I hope soon to also publish in English), I have grouped producing mines, forests, and public utilities, for taxation, apart from agricultural and city land. The assessment of the selling value for taxation I have restricted to the latter group; and have assessed the members of the former group on their annual rent. By taxing rent instead of its capitalization or selling value, the problem of future ore prices would be eliminated from the calculation.

6. *Valuation of Undeveloped Mining Land, and Summary.*—Mr. White's affirmation, "The State has no right to force an owner to sell or to name a price at which he will sell," not only makes no distinction between private property in land and private property in labor products (see references, footnotes 5 and 7), but ignores the legal basis of existing land titles, which reserves to the State the power both of taxation and of eminent domain. The condition pictured in Mr. White's concluding paragraph is certainly not a reassuring one for believers in equality of economic opportunity. It makes it evident that Michigan landlords have as much power to tyrannize over the producers, who work their lands for them, as have even their counterparts in aristocratic England.

In proposing the self-assessment of undeveloped land, I was not considering the financial desires or hopes of speculators but the general interest of the public and of the mineral industry. Neither can I share Mr. White's alarm at the danger from overproduction that may result; since the chief effect of the destruction of speculation will be the beneficent one of the opening to development of much superior land which is now idle. As the same labor and capital can produce ore at a lower cost from superior than from inferior land, there would be a consequent fall in ore prices, without lowering wages or interest; and the consumer would be correspondingly benefited without affecting any interest but the rent of the landlord, the residual legatee of production.

The reversion to the State of much unexplored mineral land, for nonpayment of taxes under self-assessment, I should consider an unmixed blessing, harmful to no interests except the anti-social ones of the speculator. The latter class appear to me as mere industrial parasites, the greatest existing obstacles to the free development of natural resources; and their dislodgment and the consequent possession by the State of large mineral areas, freely open to the prospector and operator, would offer the best offset to the apparently unlimited power over the operators now possessed by the Michigan fee owners.

[7] R. B. Brinsmade: *El Latifundismo Mexicano; su Origen y su Remedio.* Published by Secretario de Fomento, Mexico, D. F.

The Reserves of Iron Ore for the United States

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(Pittsburgh Meeting, October, 1914)

EXTENDED discussions, by inviting attention to problems affecting the conservation of natural resources, have encouraged investigations as to their sufficiency, with the general result that the more thorough the examination, the less cause appears for anxiety as to the exhaustion or serious depletion of these resources. The extent of industrial development, the large plants and the enormous volumes of materials required, when compared with conditions a few decades ago, are sufficiently startling to enlist the attention of those who "fear the pace," and yet in but few instances has real solicitude for the future been warranted by a knowledge of true conditions.

In a former communication, presented five years ago,[1] I endeavored to emphasize the fact that true conservation represented the best utilization of reserves and not the locking up of resources for future use; a conviction which has become more pronounced as the subject was investigated. Prophecies of the early exhaustion of mineral resources have raised the question as to our permanent reliance upon these, and encouraged inquiries which have in most cases developed reassuring conditions. But the fact that the future appears less dark than prognosticated cannot excuse wastefulness or improvidence; and detailed study which demonstrates an apparent abundance of natural resources should encourage their best application.

In making inventories of resources one is liable to confine conclusions as to availability upon conditions in the past and present circumstances, or to formulate data for the future upon methods which are rapidly being displaced. In reserves which are not reproductive, such as minerals, the possibilities of exhausting the supply are startling if estimates assume that future applications will follow methods which were less economical than those now employed, or to be used, and we dare not claim that the limit of advanced practice has been attained.

Examinations and estimates of iron-ore reserves may be taken as illustrative to determine whether an ample supply exists in our own country, or judge of the possibilities of liberal accretions which may be obtained from deposits of phenomenal character or size located elsewhere.

[1] The Conservation of Natural Resources, *Trans.*, xl, 412 to 418 (1909).

The prevalent method of estimating iron-ore resources has been to consider them as applicable to the present status and location of industrial development, to methods of utilization in vogue, and to processes which are accepted as the best in the light of existing knowledge.

A retrospect of the history of iron production in the United States demonstrates that the location of industrial centers dependent upon the smelting of iron ores has materially changed, and that many blast furnaces, formerly relying upon local ores, now obtain their supplies from distant points. The abundance and quality of mineral has caused the greatest development during the past 20 or 30 years at localities to which ores from the Lake Superior region have been transported from rail to vessel and again from vessel to rail to reach smelting plants; but other industries have been created to supply a demand in sections of the country which several decades ago offered but a limited market for manufactured products.

As population increases, as towns and cities grow, as railroads are extended in various parts of the country more or less sparsely inhabited, as water transportation is cheapened, and as requirements from other countries may make drafts upon or furnish material for the products of the United States, changes in industrial centers may be expected. We, therefore, may not limit consideration of the iron-ore reserves solely as affecting the iron and steel industry as it now exists in this country, for railroad and water transportation cheapen the cost of delivering materials from distant points and facilitate the shipment of manufactured products.

Mechanical ingenuity, by displacing manual labor, admits of greatly augmented outputs and larger plants, while local industrial development is an early desire of growing communities.

Our iron-ore reserves are to be estimated as supplying our present industries and also as encouraging the development of new enterprises in a country of vast area, rapidly increasing in population and in markets for iron and steel products.

The history of the Institute, as set forth in its *Transactions*, shows how the centers of production and consumption have shifted, and illustrates changes which may be expected, at least in part, to be repeated.

Another factor apparently neglected is that but few iron-ore mines have been worked to exhaustion, for inquiries covering several years failed to develop any considerable number of iron-ore mines which could be classed as worked out. There are mines which have been abandoned because of accident, or by reason of apparent inferiority of the mineral to other ores recently obtainable.

There are instances of mines flooded because of inefficient pumping apparatus which have ceased operations, and of others which have been wrought without modern labor-saving appliances and were not economically exploited, but some of which may be revived and made productive

by utilizing equipment of modern design and capacity. Other mines lie dormant by reason of some constituent, which is or has been considered as deleterious, or because existing transportation facilities do not encourage immediate development, and limited financial ability of some who attempted to operate iron mines or blast furnaces, is also blamable for inactive deposits.

But a careful review of the iron-ore deposits of the United States, which have been or are wrought, shows remarkably few instances where the deposit has been "bottomed" or followed to its lateral termini; on the contrary, the testimony of those operating iron-ore mines, proves that in the majority of instances, the more a deposit is exploited, the greater are the apparent reserves. After nearly 60 years of activity, new "finds" of iron ore are reported from the Marquette range in Michigan, and the same is true of the Menominee range, which was opened 20 years subsequent to the Marquette range.

The Gogebic range of Michigan and Wisconsin, after a history of 30 years, has been demonstrated to possess enormous deposits of desirable iron ores below what had previously been considered a dike defining the depth of orebodies. And later discoveries between the Gogebic and Eastern ranges have proven large quantities of iron ore. In Minnesota, which 30 years ago supplied no iron ore but has grown to be the largest known producer, ore is now obtained in sections which until lately were considered beyond the productive limits; while explorations are developing additional ore supplies. One hard-ore property, which had been a large producer but abandoned because of its apparent exhaustion, is still contributing to the ore supply and the deposit has been proven to extend into the adjoining properties, where it is successfully wrought.

The magnetic iron mines of the Lake Champlain district, which have long been active, show no signs of exhaustion and new explorations are exposing orebodies whose magnitudes have been estimated in hundreds of millions of tons. The Cornwall ore banks in Pennsylvania, which have been supplying ore for over 170 years, have lately been equipped on a scale demonstrating that the owners are satisfied of liberal reserves for many years to come. Active mining, which has been prosecuted for about 40 years in the Birmingham district in Alabama, does not suggest exhaustion, but rather directs attention to the probabilities of a liberal supply of ore for the future. Iron ores have been determined as existing in liberal amounts either by exploration or by more or less continued exploitation in the eastern and central parts of the country, also in the Rocky Mountain region, and the territory to the west. In Colorado an important iron and steel industry has been maintained for more than three decades upon ores mined from deposits, some of which are of known magnitude, in the Rocky Mountain region.

The litigation between the National Government and the United

States Steel Corporation brought forth sworn testimony from experts, among which were the following estimates of iron-ore reserves:

In Michigan: 170,000,000 long tons,[2] or if high-silica, low-phosphorus ores of 40 per cent. grade are included, 219,000,000 long tons.

Minnesota in 1911 was credited with 1,670,000,000 tons, but the Cuyuna being then estimated at 40,000,000 tons was subsequently increased to 400,000,000 tons. The tonnage of reserves of the Great Northern lands was estimated at 235,416,756 tons.

If all ore above a minimum of 45 per cent. of iron is considered, there would be an additional iron-ore reserve in the Lake Superior district of at least 2,000,000,000 tons, and if the minimum is 35 per cent., 70,000,000,000 tons.

The Adirondack region of New York was estimated to contain 800,000,000 tons, which would yield 300,000,000 tons of furnace ore; and for southeastern New York 60,000,000 tons of reserves were estimated.

New Jersey was stated to contain 130,000,000 tons of reserves, of which 30,000,000 tons were shipping ore, and the balance would yield 40,000,000 tons of concentrates carrying 60 per cent. of iron.

In Alabama, the Birmingham district was estimated as having 1,300,000,000 tons, the Russellville district 100,000,000 tons, and eastern Alabama not over 25,000,000 tons.

Virginia was credited with 100,000,000 tons and Texas, in Cass and adjoining counties has 250,000,000 tons of brown ores.

To the above could be added reserves of considerable quantities in mines which have been exploited, and possibilities of known iron-ore deposits in sections of the country which have not heretofore supplied iron ores or offered a market for them.

In nearly all the political divisions of the United States iron ores of apparently desirable quality and quantity are known to exist and geological reports upon a number of important iron-ore deposits demonstrate the probabilities of liberal extensions beyond present exploitation in depth or length or both, and the working of mines has in many instances fully warranted the geologists' conclusions.

Iron-ore producing regions are generally areas of iron-bearing rocks, in which there are local concentrations carrying sufficient iron to encourage exploitation as merchantable iron ore, and the grouping of the deposits which have been or are now the sources of iron-ore supply demonstrates that the above may be considered as the rule rather than the exception, although exceptions may be recalled. Until these iron-ore bearing areas are thoroughly explored, new discoveries of workable mines may be expected, and until exploitation discloses the limits of the mineral, their capacities cannot be calculated.

[2] The Michigan Tax Commission in 1913 made new estimates of the reserves of ore, showing a total of 189,467,621 tons.

To the above may be added the foreign sources which may supplement the domestic supply of iron ores, upon some of which testimony was presented in the litigation above mentioned.

Cuba was estimated to have 3,000,000 to 6,000,000 tons of hard ore, and 3,000,000,000 to 3,400,000,000 tons of soft ores.

The Tofo mines in Chili, is a mountain of ore, which for a distance of but half way from the top was estimated to have 150,000,000 tons of 67.5 per cent. ore, and if wrought to the level of the valley would yield probably three or four times as much.

The Newfoundland iron ores were estimated at 3,250,000,000 to 5,000,000,000 tons, and in the Minas Geraes district of Brazil 7,000,000,000 to 8,000,000,000 tons of 60 per cent. iron ore were claimed.

Sweden is credited with 1,500,000,000 tons of reserves in Kiiruna and 493,000,000 tons in other deposits.

Much additional detailed information concerning the future supply of iron ore appears in the volume entitled *The Iron Ore Reserves of the World*, a part of the proceedings of the International Geological Congress held at Stockholm, Sweden, in 1910, and in published monographs on special districts or deposits, many of the estimates of which have been exceeded by later detailed reports.

Some iron industries formerly important have passed into memory, and mines which for long terms of years produced mineral lie dormant, but on the other hand new developments are noted and mining operations are on a scale of magnitude attracting world-wide attention. Industries of moderate size which these effete installations supplied have expanded and demand iron or its manufactures in large volume. The mines abandoned may not have yielded all their ore, but in the march of progress the volume of business has been greatly extended and other sources are utilized. It is not improbable that some of the idle plants and dormant mines may be revivified and that the pendulum of production which in the United States swung geographically westward may return, at least in part, to the East. Localities could be cited where iron-ore mines have been or are operating, for which continuous life may be confidently expected, and the more thoroughly they are explored, their greater possibilities may be demonstrated.

Another phase in estimates of reserves is the changing demand for ores of specific compositions, as was instanced by the apparent scarcity of ore of Bessemer grade being the incentive for a liberal development of the basic open-hearth steel industry, thus making available much mineral formerly considered under the ban and reducing the premium on Bessemer ores. Millions of tons of concentrated ore, treated to raise the iron content or eliminate undesirable constituents, are now annually used in the United States. Northern New York contributes large amounts of magnetic concentrates, Minnesota and the Southern States supply quan-

tities of washed ore, suggesting that other deposits in which the mineral is lean in iron, or which contains objectionable ingredients, may, with modern appliances, add to the supply of the country.

Heat treatment of iron ores also increases the available supply by sintering or nodulizing finely comminuted material and driving off sulphur and volatile matter.

All methods of beneficiating iron ores add to the cost of the marketable product but the high grade obtained may offset this expense by commanding increased ton rates, and the outlay necessary to improve local ores may be less than the transportation charges on others from distant mines.

Metallurgy, which has mastered many difficult problems, may be expected to overcome successfully apparently unfavorable compositions of iron ores. Copper, which has been objected to as a drawback in manipulating iron associated with this element, is now desired by some manufacturers, and the presence of this metal within certain limits even suggests a premium due to its occurrence with the iron. Objection to the use of sulphurous iron ores is less pronounced than formerly, for these are fed to furnaces whose burden is adjusted and fluxed practically to neutralize this element. Titanium, chromium and other elements which have been under the ban are accepted as component parts of iron ores for use, and phosphorus, which was classed as most objectionable, has lost much of its bad reputation, and is even purchased to add to the charge fed to some blast furnaces which smelt iron ores deficient in this element.

Most estimates of iron-ore reserves assume that the present processes of smelting and conversion are to continue. Marvelous progress has been made in perfecting these processes, quantity and quality of product being obtained at costs far below those of a few decades ago, and this advance may be expected to continue, with possibly even more radical changes in the future than have heretofore taken place. The blast furnace has been the accepted appliance for producing iron for centuries, but the plant at the present time is quite different from one of even 50 years ago, not only in the quantity produced but in the materials used, the practice followed and the resultant economies. It is possible that another 50 years may so change the smelting process as practically to eliminate the blast furnace, and in estimating the iron-ore reserves we should consider that new methods may be employed for using ores which are now considered relatively undesirable.

The production of ferric alloys of titanium, phosphorus, chromium, silicon, manganese, etc., warrants the expectation that these may grow in importance, and that some ores in which the above or other elements are prominent, may be in much demand, for in many lines of industry steels of specific composition, in which some of the rarer elements are alloyed, are sought and the production of such steels has already reached an important volume. For such purposes ores of which iron is the base, but

which are at present lightly esteemed because of other items of composition, may be sought and liberally employed.

While in the United States the electrical furnace has been primarily applied to refining steel, the possibilities of electrical smelting may command attention, especially where rich ores can be treated with electricity developed by water power or by fuel at low cost, a possibility influenced by the highly economic operation of turbo generators.

Considering the various features above enumerated and remembering how much of our mineral wealth remains to be proven, there would seem to be little ground for anxiety as to the future supply of iron ores for the United States, especially if the question as to the years for which a supply should be calculated enters into the discussion. Such an estimate should take into consideration the factors which have been above outlined and also the financial problem; for it is fairly presumable that any deposit of iron ore demands an outlay of money for its purchase or lease-hold; and for the capital invested in plant upon which interest and amortization must be provided, while the growing tendency to base taxation upon calculated reserves places on the iron miner the necessity of paying continuously the taxes upon mineral which he may not win for many years, or which may be ultimately lost by fire, flooding or other accident.

The return for the money invested in the iron and steel industry or in the mining of iron ore, will be affected by the duration of the demand for its products, and by competition which must be met from other developments or industries subsequently brought into activity. The investor will consider the extension of productive facilities elsewhere, the growth of communities and the local demand which these will make. If the ore produced is practically free from undesirable components it may have to meet competition from other ores from which these components are removed by processes of beneficiation, and the demand for metals of certain alloys may also influence the amount of mineral to be won and the use to which it is to be applied. It may then be questioned whether, in providing supplies of iron ores, one is justified in assuming that a specific deposit or series of deposits may be the dependence for an industry far in advance.

This paper could be extended by instancing individual deposits of iron ore or specific industrial installations as proof of the general conclusions presented, but if it invites attention to what may be classed errors in prognosticating reserves, or it awakens discussion upon future changes which may be expected, its purpose will have been served.

The discussion has been confined to the iron-ore reserves of the United States or those which have been estimated as available to supplement the domestic supply and while there has been no attempt to consider the iron-ore deposits of the world, it is believed the sufficiency of these may also be demonstrated.

The Need of Uniform Methods of Sampling Lake Superior Iron Ore

BY C. B. MURRAY, CLEVELAND, OHIO

(New York Meeting, February, 1914)

"OUR experience from time immemorial has been that the furnace sampler and analyst usually find one or two, and sometimes more, points less iron than the Lake Superior shippers, but, fortunately, their contracts are 'iron bound'; and, generally speaking, analyses at this end are practically useless, for we are obliged to settle on a basis figured out to a half dozen decimals, and, as a rule, without recourse in the event of flagrant discrepancies between the two analysts."

The above is an extract from a letter received by the writer of this paper from the President of an iron company, discussing the question of sampling. It so well expresses the opinion of many blast-furnace owners and managers that it seems a suitable text for a discussion concerning the need of uniform methods of sampling.

If all railroad cars were loaded the same, if all stock piles were the same shape, and if ore was received regularly each day, the problem of sampling would be comparatively simple. The conditions, however, vary widely at different places, and at the same place from day to day, and it is probably impossible to determine upon any one method of sampling which will apply at all times and in all cases. However, it would seem as though some general principles might be suggested which would tend to overcome these wide differences in samples of the same thing.

The sampling of a cargo of iron ore begins at the breast of ore in the mine. The ore is sampled from the mine cars, from the stock piles, and from the railroad cars at the mines. At the lower Lake ports the ore is practically all sampled from the boat, occasionally from railroad cars. At the furnaces, the ore is nearly all sampled from railroad cars and occasionally from stock piles.

During the last ten years, sampling at the mines has been very much improved. The question has been very generally discussed, and each year has seen some improvement. The United States Steel Corporation, in 1908, took a very decided stand in this matter. After carefully looking into the various methods of sampling used at their own mines, a committee recommended certain methods of sampling, differing somewhat at the producer's and at the consumer's end. These methods were adopted

and have since been used at all the mines and furnaces controlled by the Corporation.

These methods, or modifications of them, have been adopted by some of the other mines in the Lake Superior region, but many of the mines are as yet paying but little attention to sampling from a scientific standpoint. Last summer Benedict Crowell read a paper before the Lake Superior Mining Institute on Methods of Sampling at Lake Superior Mines. This paper brings out very clearly the need of some standard method of sampling at the mine, and suggests the details of such a method.

Prior to 1907, there were several methods of sampling in use by the Cleveland chemists in sampling iron ore at the lower Lake ports—some favoring small samples, others favoring large samples; some taking one round when the boat was half unloaded, others taking three rounds, at certain stages of the unloading. These different methods of sampling resulted in discrepancies in results, particularly if the ores were mixed.

At the beginning of the season of 1907, the Cleveland chemists adopted a standard method of sampling, which was a step in the right direction. This standard method has been broadened and improved from year to year, and, while it is not yet perfect, it has shown itself capable of giving accurate results if conscientiously carried out. The method is as follows:

A standard sample shall be taken from all cargoes, the weight of the sample varying with the size of the cargo.

The sample shall be taken with a galvanized iron scoop 3.5 in. wide, 2.25 in. long, and 1.25 in. deep, the handle 8 in. long; and with a hammer 12 in. long (the scoop holds approximately 0.5 lb.).

It shall be the aim to take an equal bulk of ore from every point selected. When a lump is encountered, a portion shall be broken off equal in bulk to a scoopful of soft ore. In sampling cargoes, no sample shall be taken from the original outside surface on account of the presence of foreign matter and an undue proportion of fines.

If the surface to be sampled has been exposed to rain or sun long enough to materially alter the amount of moisture present, the sample shall be taken about 3 in. below the surface.

In order to keep the size of samples within reasonable bounds and to gauge this size to the size of the boat, the sampler shall, on cargoes up to 4,000 tons, begin sampling at a convenient point at the bottom of the face of the ore, and shall take one standard scoopful every two scoop lengths up the face of the ore to the top, and then shall move four scoop lengths to one side of the starting point before again sampling vertically. He shall continue in this manner, keeping the above distances, around the face of the ore to the place of beginning.

On cargoes from 4,000 to 6,000 tons, he shall use the two scoop lengths for vertical distances up the face of the ore, but move six scoop lengths horizontally.

On cargoes over 6,000 tons, he shall use the two scoop lengths for vertical distances up the face of the ore, but move eight scoop lengths horizontally.

In case of split cargoes, horizontal spacing to be according to the tonnage of each individual ore as specified in the preceding paragraphs.

At convenient stages of unloading, the sample shall be mixed and quartered. This must be done each time exactly alike, by breaking down to 0.5 in., mixing and quartering twice, thus preserving the proper proportion of the whole sample.

If in the final quartering the last two quarters exceed a can full, the ore shall be quartered and one quarter rejected.

The sample may be quartered on the vessel, or may be taken to some other place suitable for that purpose. Samples must be shipped in standard cans.

Sampling of Soft Ore

The sampler shall enter any hatch and begin sampling when the unloading machines have exposed 5 or 6 ft. of the face.

The sampler shall then enter the next hatch working, and proceed to sample in the same way, and so continue in every working hatch.

The sampler shall then begin over again in the hatch in which he first started and continue the sampling in all the working hatches, provided there has been sufficient ore removed in such hatches since the faces were sampled to expose fresh ore.

The sampler shall continue this method in each hatch worked until there is less than one-tenth of the ore left.

In sampling horizontal surfaces, as in boats where scrapers are used, the sampler shall sample every two scoop lengths lengthwise of the boat, the spaces between the lines of sampling to be 4–6–8 scoop lengths, according to the tonnage, as described before.

Sampling of Hard Ore

In sampling hard lump ore, the sampler shall begin sampling and use the same spacing as defined for soft ore, using hammer lengths instead of scoop lengths. At each point sampled he shall take lump or fine ore equal to 1 cu. in. In taking this cubic inch, the sampler shall take an average from the lump ore from which the cubic inch is broken.

Moisture Sample

The moisture sample shall be taken from the standard sample in the following manner:

When as many cans of ore have been filled as the stage of unloading will permit, the lump ore shall be broken up quickly and the entire amount thoroughly mixed and flattened out into a circular pile. The pile shall

then be marked into quarters, and one of the quarters divided into two parts by a line from the center to the outside of the pile.

The whole of one of the half quarters so marked off is to be placed at once in a tightly covered receptacle to be a portion of the moisture sample. The other half of the quarter together with the whole of the opposite quarter is to be rejected.

The remainder of the pile is to be thoroughly mixed and flattened, and two opposite quarters rejected. The remainder of the ore is to be put one side for a portion of the standard sample.

By this method of procedure, one-eighth of the entire sample taken will go into the moisture sample.

At the end of the sampling, the accumulated moisture sample is to be taken out of the tightly covered receptacle, quickly mixed and flattened out, divided into eighths and enough eighths taken to fill the standard moisture can.

In case of hard ores or small tonnages of soft ore, the proportion set aside for the moisture sample shall be increased so that at least one can of ore shall remain for the final moisture sample.

The moisture determination is made by drying the whole of the final sample at a temperature not over 212° F. until there is no loss in weight. The loss of weight divided by the amount of ore taken will give the percentage of moisture.

Variation in Methods of Sampling

It is at the various furnace plants where Lake Superior iron ores are used that we find the widest variation in methods of sampling.

Some months ago the writer sent out a circular letter to about 30 furnace plants, asking for a description of their method of sampling. It seems worth while to give the replies from a number of these plants and from these replies a conclusion can be drawn as to the need of some uniform method of sampling at the furnaces.

Plant A: "We send a man out with a bucket and take a small portion out of six different places in each car. We do this on every car that we get out of a cargo, and from a cargo of about 7,000 tons we get about enough out of it for a sample to fill a 12-quart can."

Plant B: "Cars as a rule are loaded with two piles of ore, approximately 25 tons at either end of the car. The sampler cross-sections roughly each pile and takes four small shovelfuls off each, or eight shovelfuls from each car. These car samples are collected in one large pile until the entire cargo is received. The entire pile is passed through a crusher and crushed to 3/4-in. screen. Next the entire sample is coned and quartered down to a bucketful, or approximately 15 lb., which is taken to the laboratory."

Plant C: (The description of this method was preceded by a mathematical demonstration of finding the volume of a cone 4 ft. high and 6 ft. base and showing that one-half the volume of the cone is in the lower 0.826 ft. of the height.) "The ore in the cars is usually in three piles. These piles are cone-shaped, with apex of the cone

rounded off. The piles are sampled as follows: One sample taken near the floor of car, one sample taken near the top of pile, and another sample taken at one-third the height of pile. If at the first place the ore is all lump, a measured trowel of lumps broken from the different sized lumps is taken. Each pile in the car is sampled in the same manner, or nine trowels full taken to a car."

Plant D: "When a sample is desired, it is usually taken from the filling barrow, using a hammer and a small shovel so as to keep the proportion of fine and coarse ore as it is. This sample averages about 20 lb."

Plant E: "We sample from 30 to 40 cars in each cargo as follows: When the ore is loaded in the car in one pile we take a scoop at each corner about 2 ft. up from the bottom and about the same distance in from the sides. Then a scoop on each side at the top toward the sides. The lump is sampled according to the amount in each car, judging as near as possible the proportion of lump to fine. When the ore is loaded in two piles the sample is taken the same way, except that we take one scoop at the top."

Plant F: "The sampling from cars, from five to seven buckets full, averaging 40 to 50 lb. to the bucket, are taken from each cargo from cars and mixed and quartered down to about 30 lb. We sample across the car every 2 ft., taking a scoopful, and then lengthwise, measuring 5 ft. for each transverse line of sampling."

Plant G: "Our method of sampling, we presume, is similar to that of the average small furnace plant; that is to say, with a hammer and trowel, we go over the cars, remove the surface, so as to avoid any contamination of foreign matter, and when possible digging a hole down through to the bottom of the car, and estimate the percentage of coarse and fine therein and take samples accordingly."

Plant H: "Our method of sampling has been to have the man go over the car and take samples from all parts of the car, pulverize and thoroughly mix same."

Plant I: "In view of the fact that we mine our own ore and the question of sampling same at the furnaces is not a very important matter, I do not think we would be interested in adopting any uniform method."

Plant J: "On arrival of a train load of ore a sampler goes over each car pretty carefully with the aid of a hammer and small shovel, and after selecting a few pounds from 8 or 10 cars, the material is taken to the laboratory and put through a regular course of quartering and simmering down to the desired quantity."

Plant K: "We have no regular method of sampling."

Plant L: "Sampling at our furnaces is all done as cars are unloaded. A bucketful of ore is taken from each car, thereafter usual laboratory method follows."

Plant M: "In sampling cars, we zig-zag across the top of the cars, taking samples at equal distances. Wherever we have the chance, we drop the bottom of the cars and then sample the two remaining ends of the cars, as this gives us a sloping surface from top to bottom of cars."

Plant N: "Our practice in taking samples from cars is to lay off the surface with an imaginary line in squares and to take as uniform a sample from each as possible."

Plant O: "Each car is sampled at 40 different places, due care being taken that the lump ore represents the same ratio in the sample as exists in the ore itself. The size usually taken from the lump is that of an English walnut, although this is governed by the size of the lumps, and the fine ore is sampled so as to be in proper proportion. In case of gangue, a piece is taken, the amount depending upon the relative amount present."

Plant P: "In sampling ore from cars we use a bucket. Starting at one end of the car or the left side, we go the entire length of the car, breaking the lump and taking some of the inside and outside, also digging down three times and getting the fines. Then sample the center and other side of the car, taking samples 3 ft. apart. We usually have 3 quarts to represent a car."

Most of the other replies showed that an attempt was being made to sample each car in a uniform way; but in all of the methods, the amount of lump and fine taken was left more or less to the judgment of the sampler.

And yet the results of these samplings are compared with the samples of the cargo taken at lower Lake ports, where two or more men have perhaps sampled continually by the standard method for 12 to 18 hr. If the analyses vary widely there is a complaint, and usually the furnace manager insists that his sample is the correct one. It is remarkable that there are not wider differences in the analyses of the two samples, and until more uniformity in taking samples at the furnaces is adopted there will be these wide variations.

Theoretically, sampling is capable of definite mathematical demonstration. In practice, however, we are confronted with some difficulties in attempting to sample by mathematical formulæ.

First, piles of ore vary widely in size and shape. While they may, in general, approach a cone shape, it is rarely that they come near enough to this definite shape to use the formula of cone-volumes in sampling.

Secondly, sampling as a rule is done by uneducated men, to whom formulæ are unknown, and whose judgment as to percentage of lump and fine ore in sight is not infrequently biased by the known wishes of their superiors. Again, the judgment of any two men is liable to vary widely in estimating the amount of lump and fine in a pile of ore; and, furthermore, the proportion of lump and fine in sight may be, and often is, quite different from the proportion in the whole pile.

In my opinion, a mechanical sample is much to be preferred to a judgment sample, especially when the sample represents a considerable amount of ore. Ore as received in cars at furnace plants has been loaded by a grab bucket, in nearly all cases. This method of loading, of itself, tends to mix the ore so that a proper sample from the surface is fairly representative of the whole car.

The jolting motion of cars in transit may perhaps, in some cases, cause a rearrangement of the lump and fine and bring a little more lump to the surface than is the proper proportion. But in sampling a number of cars from the same cargo I do not believe that this will make any material difference.

A Suggested Method for Sampling Cars

I suggest the following method for sampling cars, which I believe could be used in nearly all cases to get a representative sample, whether there be one, two, or three piles of ore in the car. Starting at a point 1 ft. from the end of the car, move in a straight line lengthwise through the middle of the car over the piles of ore, taking equal bulks of ore every 3 ft. Measure these distances, and at the designated point dig about 4 in. below the surface and take a measured bulk of either lump or fine, which-

ever is encountered. A scoop holding about 0.5 lb. of ore could be used, with a handle about 12 in. long, and the scoop may be used for measuring the distances. Or a measuring stick of the right length could be used, without inconvenience to the sampler. This method of sampling could be elaborated, if thought necessary, by going across the car twice more on lines about two-thirds the distance from the center of the car to the side of the car. This, of course, touches the ore in three times as many places, and in the case of mixed ore would tend to give a more representative sample. In a majority of cases, however, in my opinion, it would not be necessary.

The most general objection raised at furnace plants to sampling by measuring distances and taking samples from a number of places is that they do not have the time. At the smaller plants, the sampling is generally done by a laborer who has other work to do, either about the plant or in the laboratory, such as crushing or cleaning up. This objection brings up the point of the value of sampling at the furnace. Is it of any particular value, and if it is, why should it not be done in some reliable way?

I take it that at most plants the ore is sampled chiefly for two reasons: first, to check up the Cleveland chemists; and secondly, to use the analysis of the sampling to burden the furnace. In either case, accuracy is wanted. Why not, then, make a business of sampling?

Analytical results can be redetermined, and if in error can be corrected. After the sample is taken, however, in a majority of cases, no resampling can be done in a satisfactory manner. It would seem then a very self-evident fact that the sampling should be done right, and to do it right requires time and attention, both of which cost money.

I am making this plea for good sampling and suggesting a method of taking samples with the hope that I can bring out some ideas along these lines and all—mines, Cleveland chemists, and furnace interests—get closer together and work more in harmony.

As soon as we can do this, such opinions as are expressed in the opening paragraph of this paper will be removed.

Discussion

William Kelly, Vulcan, Mich.—I would like to ask the author a question. A good deal of the Lake Superior ore is made up of mixtures from various mines, of which the product differs considerably. I know of one case where ore running about 52 to 53 per cent. in iron, with considerable lime and magnesia, a self-fluxing ore, is mixed with ore running 61 per cent. in iron, in which there is very little lime and magnesia.

Now, in this method of sampling in the vessels I would like to know whether a correct average result can be obtained if all the low ore is

in the bottom and the high ore on top, or the reverse. Do you not get in this method of sampling more of the top ore than of the bottom ore? I want to disclaim, of course, any intimation that any one in the Lake Superior region would attempt to put the high ore on top in order to get a sample that would run above the average. That would not only be dishonest, but in the long run of course it would not be to the advantage of the shipper. Still, it has been suggested that in this method of sampling there is more of the top ore obtained than of the middle and bottom of the cargo, and I would like to ask the author whether such is the case.

C. B. Murray—Mr. Kelly brings up a question which is often asked by those not familiar with ore sampling at lower Lake ports. Ores are mixed, more or less, in cargoes, and unless the various grades are represented in the sample the complete sample is not representative of the cargo.

The standard method of sampling is particularly adapted to such cases, and when properly carried out takes the right proportion from the top, middle, and bottom parts of the cargo. The ore, as it lies in the boat, approaches a cone shape under each hatch. The sampler starts sampling in each hatch when 5 or 6 ft. of the ore face is exposed. As the ore is worked out, and fresh surfaces and larger areas are exposed, the sampler continues work, taking equal volumes of ore at regular, measured intervals.

William Kelly.—The thought that was in my mind was, that as you take out the ore from a pile, the top ore falls down all the time.

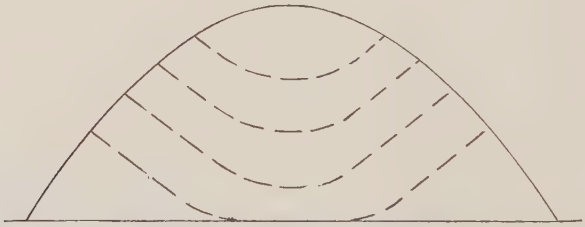
J. W. Richards, So. Bethlehem, Pa.—Then that would be counteracted, as I conceive it, Mr. Murray, by the fact that the cone being made larger, you take more of the lower layers as the cone increases in size.

C. B. Murray.—Yes, that is the idea. As the areas exposed become larger, more places are touched and more ore is taken, but the area represented by each sample remains the same. Some of the top ore may slide down the exposed surface when the grabs begin working, but as the area to be sampled increases in size, there is less tendency for this ore to slide down.

This can, perhaps, be best shown by a cross-sectional sketch of a hatch (Fig. 1).

Ralph H. Sweetser, Easton, Pa.—This paper by Mr. Murray has called attention to a detail of the iron business that is very important, not only from the commercial side, but also from the metallurgical side.

I think one of the biggest changes in blast-furnace practice in the last several years has been in the character of Lake ores shipped as Mesabi ores. Blast-furnace men have the greatest trouble in dealing with soft, fine Mesabi ores. The ores put on the market at the present time do not have these characteristics as much as they used to have, and the trouble which the furnace men have now is to work an ore which is under one name, but is really a *mixture* of two ores. The Mesabis will have, most of them, a very fine ore mixed with a great many lumps. Now, when these lumps and fines are loaded over the side of the vessel, there will be a segregation of the fines and lumps, and the samplers can always tell over which side of the boat the ore was loaded, as the lumps will go to the far side and the fines stay on the near side.



Broken lines designate surfaces at various stages of unloading.
FIG. 1.—CROSS-SECTION OF HATCH.

In one particular case we had a shipment of ore, in which the ore was mixed, fines and lumps, in the proportion, I think, of about 15 to 20 per cent. of lumps, and the rest were fines. In this particular ore, the lumps carried a great deal more manganese than the fines did. We could not make the results in the furnace check up with the analysis of the ores. That was the case in one hatch of a 30-hatch boat. I think the hatch was No. 21, and the total depth of that ore was 16.5 ft. The boat was about 42 ft. wide inside.

As Mr. Murray has explained, as they begin to unload the boat, the ore sampler will take a sample near the top part. The sampler will make his rounds, and it used to be the practice to make a round when they were one-half way down in the cargo, and then another sampling when they had gone three-quarters way down into the cargo.

This slide (referring to slide on screen, not here shown) would not give a fair sample. There is the ore, as it appeared in hatch No. 7. That is 5 ft. 3 in. deep in the center, which is just about one-third of the total height of the cone of ore which was in that hatch. The lumps are about one-third the way across. If you take a cone of ore like that, nearly one-half of the total weight of the ore would be in one-third of the height, and in order to get proper samples you would have to take as

many samples in the lower one-third as in the upper two-thirds. If the character of the ore in the lower one-third is different from that in the upper two-thirds, you do not get a fair sample of the ore. In sampling the ore, as found in a boat, it is the rule to take a scoopful every so many scoop lengths. If they came to a lump, it has been a rule in the past to take a cubic inch of lump—the scoop will hold 9.9 cu. in. of content—and if you take only a cubic inch of lump you are not giving the lump the same attention as the fines, and in the case of most Lake ores the fines are richer, and therefore you will get a higher result in the sample for that cargo.

I have been surprised to find on the Atlantic seaboard in some of the ores coming in that the fines are leaner than the lumps, and that if you do not get the right proportion in the sampling you get a lower result than the ore actually contains.

I think that the lumps should be sampled close to the sides of the vessel. There are two or three things that prevent this from being done. Those who have seen the sampling of the boats know that just as soon as they "clam out" most of the ore a man will start cleaning the side decks, and will sweep the dust over on to the ore. It is the rule of the samplers that they will not take samples after the dust has been swept on to the ore —it may be dust or it may be coal. Therefore, as a rule, the lower edge of the piles will not be sampled, and you do not get a true sample. This is because you do not get the lumps. We found in certain lumps of one ore that there were different minerals from the general run of the ore. In one case we found lumps which would contain 21 per cent. of iron, 21 per cent. of silica, and 21 per cent. of manganese, and we would question why we had such varying results in pig iron made from that ore. If we ran to lumps, the manganese went up. If we ran to fines, the manganese would run down. Whatever is decided upon as a practical method of sampling cargoes at the Lake ports would apply also to the sampling of cargoes at the Atlantic ports, because there is the same condition of loading obtaining in both places.

C. B. Murray.—The question Mr. Sweetser has brought up, about getting the proper proportions of lump and fine, is a very vital one. The lump and fine in many ores vary widely; in some the lump is richer, and in others is poorer than the fine ore. Here again the condition is ideal for using the standard method of sampling, and if the method is carried out according to instructions the sample will be representative even under the conditions which Mr. Sweetser illustrates.

It is generally the case that when samples are not representative the method has not been correctly carried out. In the case of the sampling referred to by Mr. Sweetser, the area exposed during the last stage of sampling was so much larger than that exposed during the previous

stages that the sampler would, undoubtedly, get fully as much ore as he had previously accumulated in his sample from that hatch, and if he properly measured his intervals he would take a proper proportion of the lump in the wings of the boat.

Mr. Sweetser is mistaken regarding the sampler's instructions as to taking lump ore. In sampling a cargo of all hard ore, the instructions are, "At each point sampled, take lump or fine ore equal to 1 cu. in."; but in sampling soft ore, which often contains lump, such a cargo as I understand Mr. Sweetser refers to, the instructions are, "When a lump is encountered, a portion shall be broken off equal in bulk to a scoopful of soft ore."

If, in this particular cargo, the sampler did not do this, then a correct sample was not obtained, but it was not because of the fact that the standard method was not correct, but because it was not carried out.

A. J. Ennis, Philadelphia, Pa. (communication to the Secretary*).—From my experience in the Eastern section of this country, where there are official iron-ore samplers entirely competent to perform judgment work, perhaps my appreciation of the Western dilemma, where the West doesn't pay sufficient wages to attract the services of judgment samplers and hence must reduce the operation to mechanical labor without judgment, is not sympathetic. Neither do I fall into line with Mr. Murray's sampling suggestions for the manikin sampler. For example, sampling off the long sides of the triangle cannot do the material justice. Samples should be taken from all sides. To limit sampling to the long side is to become "a one-piece-of-rock man," as it were, filling the scoop with rock where there is but one piece in sight. I agree with Mr. Murray's statement to the effect that no mathematical formula can be applied to the sampling of iron ore, whether from stock piles, cars, or ship's holds. The principle of ore sampling is, in its entirety, proportion, and for this judgment is primarily necessary. The conspicuous lack of judgment is everywhere responsible for incompetent sampling at blast-furnaces. The small-scoop practice might produce fair average results in soft fine ores, but its size—$3\frac{1}{2}$ in. wide, $2\frac{1}{4}$ in. long, and $1\frac{1}{4}$ in. deep—would debar its usefulness on hard-ore mixtures ranging from richest lump to gangue or rock, the size of scoop being insufficient to take care of lumps 6 to 15 in. in size, including surrounding ore. A scoop at least of shovel size would be necessary to the taking of *all* sizes in given spots and places. This would, of course, importantly increase the bulk of the total sample and necessitate more men to handle it. In sampling hard ores Mr. Murray reverses himself and really leaves the matter to the sampler's judgment when he allows him to take an average

* Received Jan. 17, 1914.

of the lumps from which the cubes of ore are broken. This would occur at hundreds of points throughout the cargo and constitute a judgment result on the part of the men whom he says lack the quality of judgment.

In my opinion, sampling from boats is to be roundly condemned, forasmuch as there will always be considerable unsampled ore rolling to the vessel's sides, and the fact seems to be ignored that approximately one-half of the cargo is in the lower one-third of the base of the pile. As I understand the method of Lake ore sampling, this lower third is left untouched, hence it would appear that about one-half of the cargo remains unsampled unless, peradventure, samplers have gone into said lower third of the pyramid.

The present method of sampling iron ore from cars at the docks of New York, Philadelphia, and Baltimore has satisfactorily stood the test of more than a third of a century, and pioneer samplers of 30 years ago are still on the job fortified by this long period of experience. In my opinion, it is not within the ability of man to produce for the sampler a more thorough exposure of ore than comes to pass when bucketful after bucketful of it is dumped on to cars under the eye and observation of the competent Eastern sampler inspecting its flow and accurately sampling therefrom.

L. S. Austin, Salt Lake City, Utah (communication to the Secretary*).—In July and August, 1908, I spent six weeks observing the sampling and handling of iron ore from the Mesabi ore deposits to the furnaces at South Chicago, Gary, Cleveland, and Pittsburgh. In reading Mr. Murray's paper on the subject of uniform methods of sampling, I do not find that conditions have changed at the receiving end since this investigation. I am, in consequence, of the opinion that the shipping companies are quite right in insisting upon settlements on their samples. True, there are discrepancies between the mine and the furnace samples, but the latter are not and cannot be authoritative, since they are too imperfectly and cheaply taken, and to take proper furnace samples would cost more money than the smelting companies have heretofore been willing to spend.

I will describe the methods of sampling iron ore on the Mesabi range, and compare them with those pursued at the docks at lower-lake points and at the furnaces.

The sampling of the ore on the range has been made by taking portions uniformly over the top surface of the car load. This is described in a pamphlet issued by the Chemists' Committee of the U. S. Steel Corporation on *The Method for the Commercial Sampling and Analysis of Iron Ores.* This is done immediately after loading, so that the ore may be assigned to the proper pockets at the ore docks. The united

* Received Feb. 11, 1914.

sample of a "drag" of 10 cars is taken at a time, and it is evident that an average sample of this unit can be counted on, since numerous points over all the cars are taken and at uniform distances apart. Wherever in the sampling a lump of ore is encountered this is broken and a proper proportion of it is taken. These "drags," of 8 to 12 cars but averaging

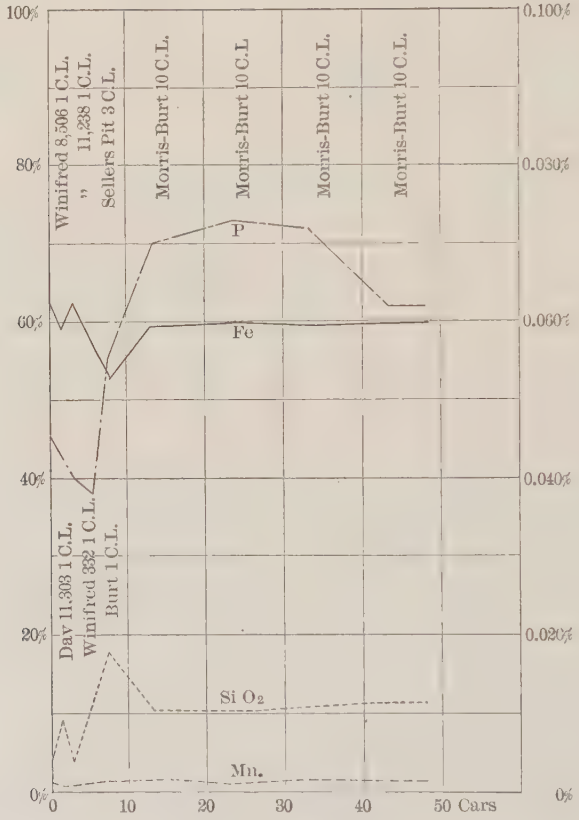


Fig. 2.—Train No. 334.

10 car loads, are made up in train loads of 50 cars and dispatched to the terminal yards, where the cars are assembled into "blocks."

The weighing is done over automatic scales, which register the weight while the train is moving slowly over the scale platform. The car number and the name of the mine are put on the registering tape, and a duplicate record is also made. A great deal of pains is taken to examine and keep the scales in repair, to check the railroad weights, and to compare them with those of a railroad scale of the ordinary type.

Loss of Weight before Weighing.—While in theory the sample of a car is to be taken at the time of weighing, the time between the taking

of this sample and the weighing is brief, and in the solid body in which the ore exists in the car the drying-out would be trifling. This idea is corroborated by the statement made me that trial car loads set out and weighed from time to time have shown an inappreciable loss of weight.

Loss of Weight.—Ore may be spilled from the cars before weighing, but this can at any time be gathered up and loaded. Ore shakes down

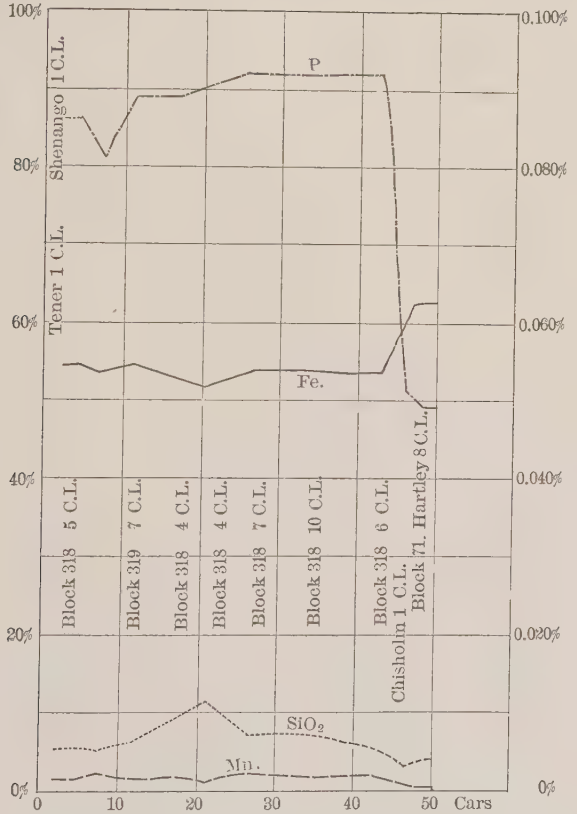


Fig. 3.—Train No. 340.

in transit, and therefore no spilling occurs after weighing until the docks are reached. Here, when discharging into the vessel, some ore escapes over the edges of the chutes into the water, but as compared with the whole bulk this does not amount to much. From the railroad weights a deduction of 1 per cent. is made for contingent losses, such as spilling and moisture loss.

Make-up of Train Loads.—The content of any train load is computed on the basis that each drag of 10 cars is uniform. Figs. 2, 3, 4, and 5

represent by graphic curves the make-up and composition of four train loads, from which the variations can be seen. At the terminal yards these trains are broken up under orders from the shipper, who is at the mine; the ores for a given cargo are selected from them, and these cars are sent to the docks.

Selection of Cars for Cargoes.—Upon examining the figures it can be seen which of the drags are Bessemer iron, for example, to go out as a Bessemer cargo. We find that for this grade the line for

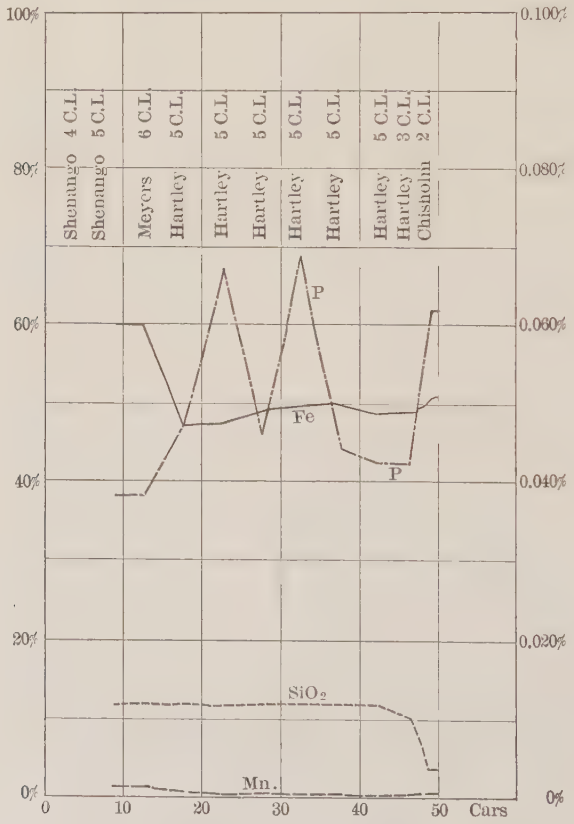


Fig. 4.—Train No. 349.

phosphorus must fall a certain distance below the line for iron. In Fig. 2 the first five carloads, and in Fig. 4 the six carloads of Meyers ore, belong in this category, with such a margin of safety that three carloads of Hartley Block 71 can be added. This ore is of a grade called "Group 2," whose requirements are that it is to contain 60.5 per cent. iron, 0.048 to 0.050 per cent. phosphorus, 5.25 per cent. silica, and 0.69 per cent. or less manganese, all based on dry weight. This corresponds to 55 per cent. iron and 0.45 per cent. phosphorus "natural";

that is, the ore as mined and containing its natural moisture, which may vary from 10 to 15 per cent.

Making up Blocks.—Upon notification that a certain vessel is to arrive at the docks, the shipper prepares a sheet called a "block"; that is, a selection of cars at the yard which will make a certain grade or "group." He wires orders for the terminal yard to send down to the docks certain cars, picked out from the train loads as above specified, and conforming to the requirements of the group which is to be shipped.

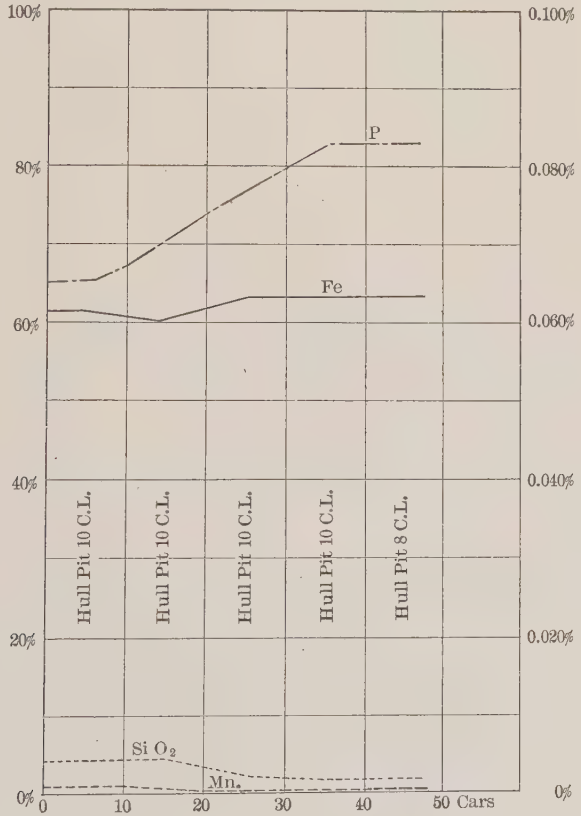


FIG. 5.—TRAIN NO. 350.

Figs. 6 and 7 are two blocks, graphically represented, one for the boat "Clemson," and one for the boat "Henry Phipps." Block 156 for the "Clemson" consists of a portion of trains 330, 340, 342, trains 343 and 344, a special train load of 390 tons of drags 140 to 151, inclusive, and "holds" 18 to 21, inclusive, being cars that have been held over and not sent down with former train loads, also 110 tons belonging to block 166, which had been put into a pocket at the docks, and which had been held until now for shipment. Proportions of older blocks (called "holds")

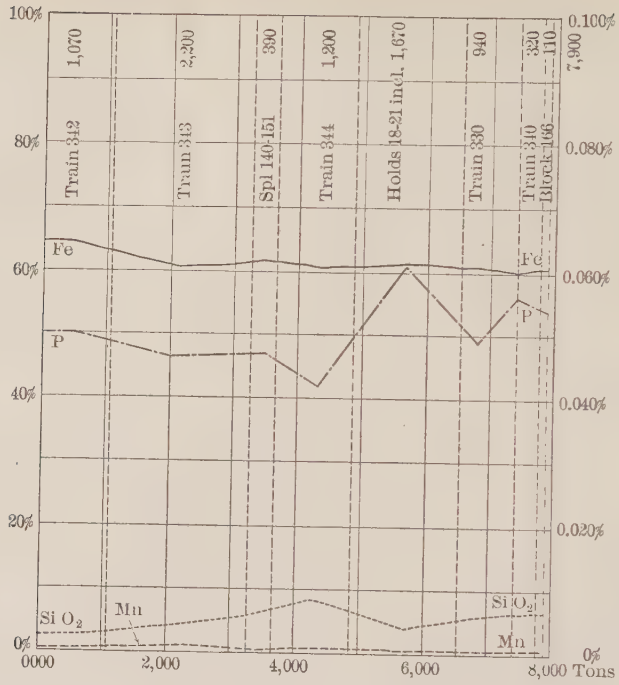


FIG. 6.—BOAT "CLEMSON." TONNAGE 7,900. BLOCK 156.

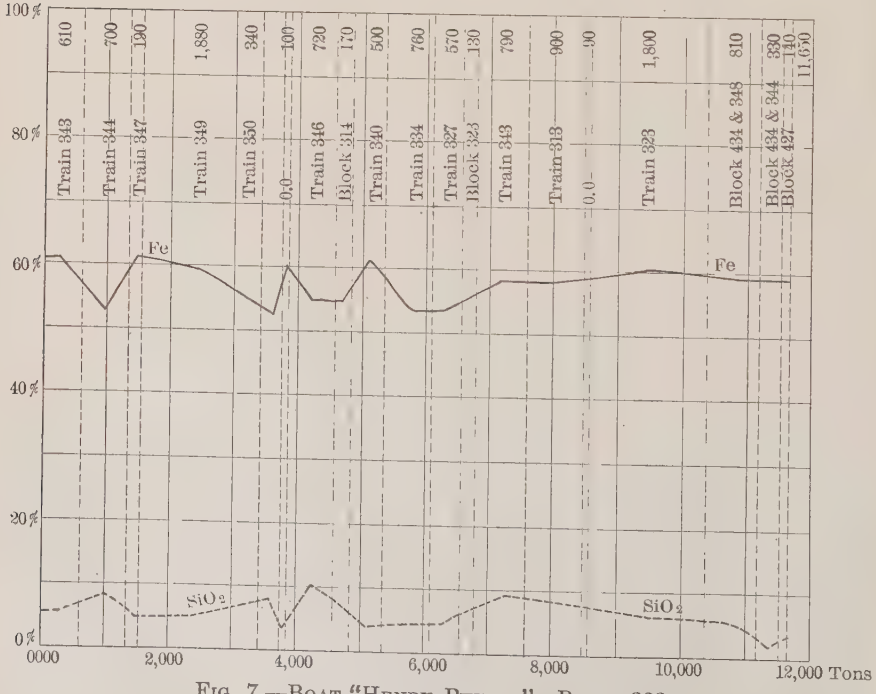


FIG. 7.—BOAT "HENRY PHIPPS." BLOCK 323.

found not to be needed for a given cargo are also held in cars at the yard, and are put into later blocks, as shown in Fig. 7. In Fig. 6 it

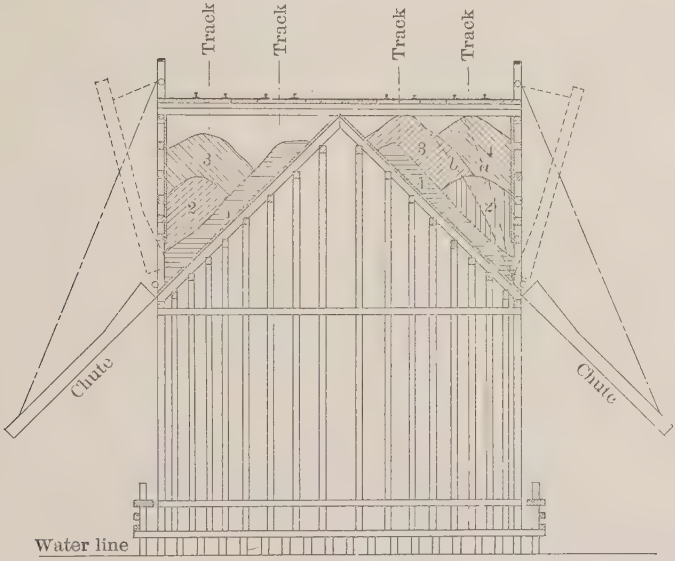


FIG. 8.—CROSS-SECTION OF DOCK AND ORE CHUTES.

will be seen that the elements vary as between different train loads but the average will conform to the requirements of the group which is to be

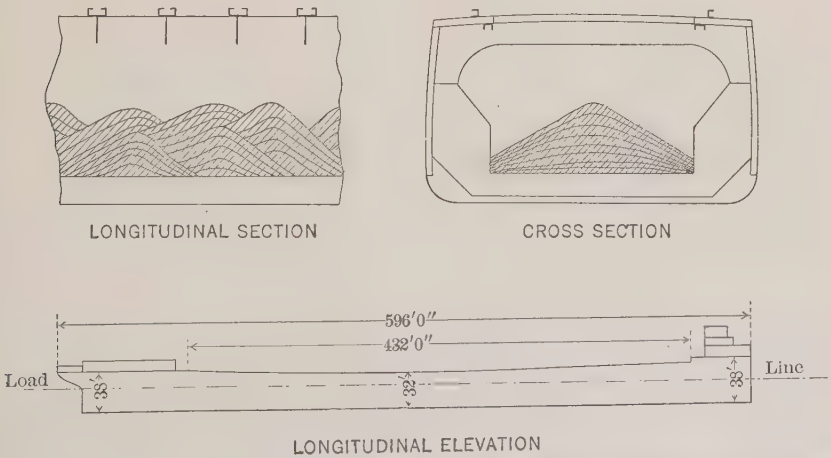


FIG. 9.—SECTIONS AND ELEVATION OF ORE BOAT, SHOWING POSITION OF ORE IN THE HOLD.

made up. Thus the "holds" 18 to 21, and train 340, would be too high in phosphorus were it not for the fact that the rest of the cargo is lower

than the requirements in that element, and can stand the addition of these holds. In Fig. 7 the specifications of Group 3 are fulfilled, the average conforming to them, though the variations in silica are such that, were it to go to the furnace no better mixed than here shown, trouble would result. To lessen this difficulty, orders are given by the shipper to the dock superintendent to scatter or distribute "off lots" through the pockets selected for holding the block in readiness for the

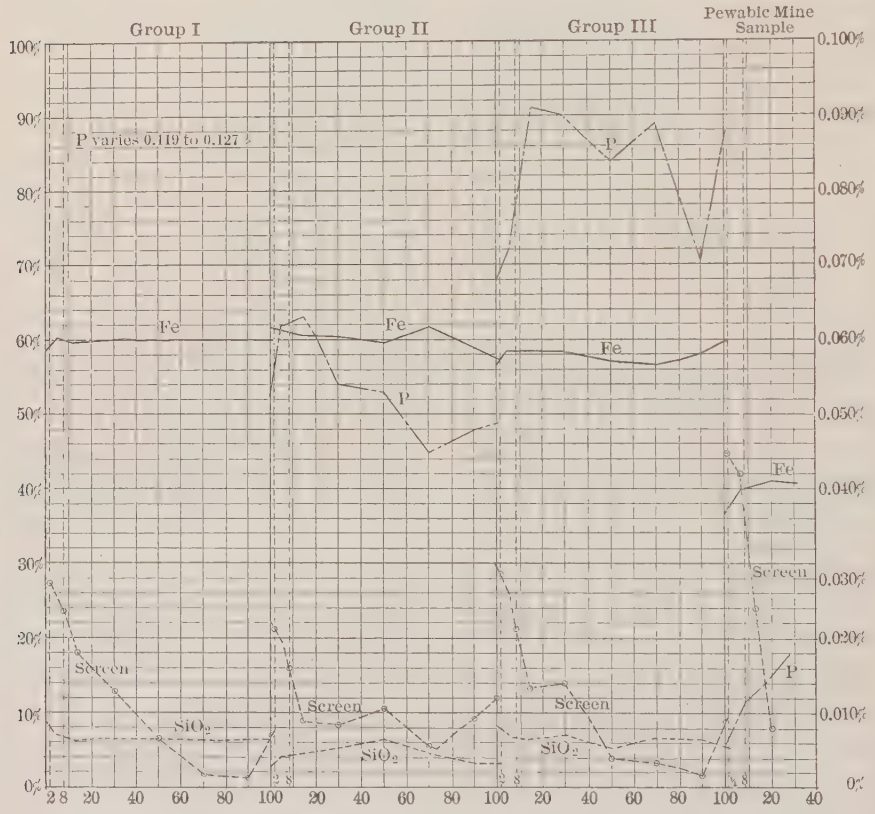


FIG. 10.—SCREEN ANALYSIS AND CONTENTS OF ORE.

intended vessel; and in addition, even to put this in the middle of the pocket, so that it will be evenly mixed with the rest.

The Ore Pockets and Dock.—Fig. 8 is a cross-section of a dock with the ore pockets and the hinged chutes for delivering the ore into the hold of the vessel. The position of the successive car loads is shown, with the order in which they fall into the pocket. These chutes are let down when the vessel is in position, the chute gate is opened, and the ore slides into the vessel. The boat itself has numerous hatches, large boats having 36 at 12 ft. center to center, so that ore may be filled into as many

as 20 hatches at a time. Commonly the force of loaders is not sufficient to keep so many going at once. The ore is poked by the loaders both at the gates and from above. It comes away first at the front, then at the back, as shown by the broken lines *a* and *b* in the right-hand pocket, cars Nos. 1 and 3 (Fig. 8) being the last to leave the pocket. Still, if two or three grades of ore go into a pocket, some portions on top accom-

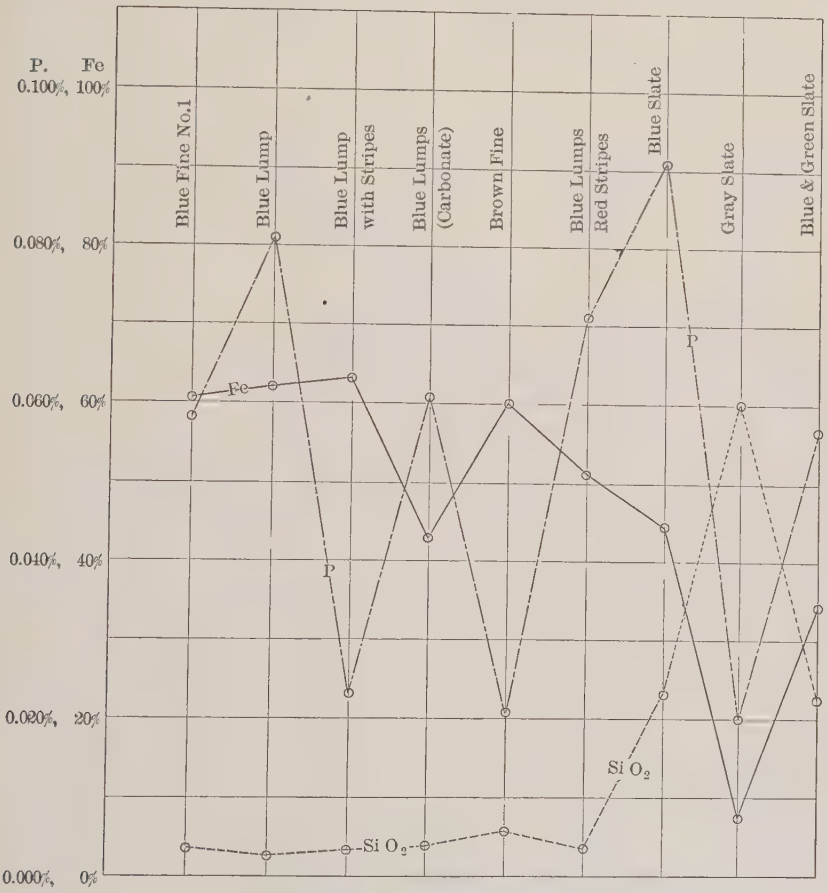


FIG. 11.—ANALYSIS OF IRON-ORE LUMPS.

pany that below, so that a good deal of mixing results. We may therefore see ore of different colors, belonging to one or other of the car loads of a pocket, sliding into the hold of the vessel. Viewed from above these layers appear to separate as they shoot downward, but they are not deep. There is a segregation of lumps and fine, the coarser ore rolling to the sides or borders of the heap forming in the hold, the fines keeping to the center or apex. This is shown in a cross-section of a 600-ft. boat, Fig. 9.

The same figure shows the ore layers in longitudinal section, and below is a longitudinal elevation of a 600-ft. ore boat.

Lumps and Fine.—Lump ore is often of the same grade as the fine; at other times, as indicated by Fig. 10, it is of lower grade. Thus in the Pewabic mine sample of that figure the oversize of $\frac{3}{4}$-in. mesh (45 per cent. of the whole) contains Fe, 37 per cent.; the size between 16 and 20 mesh (8 per cent. of the total) contains Fe, 41 per cent.; and this lump ore is low in grade as compared with the regular shipping ore. Fig. 11 shows still greater variations of lumps as picked from a cargo of the boat "Corona" in 1907. Here three kinds, blue fine No. 1, blue lump ore, and blue lump ore with stripes, carry over 60 per cent. Fe, while the three samples of slate vary from 45 to 8 per cent. Fe. Evidently such low-grade lumps, if they found their way into the small samples so often

Steamer Lynch, Load 205 Cars 11739 Tons
Av.per Car 44.29 Tons
Forward
Draft 19'10"
20'3" Aft.
Hatches 4 8 12 16 20 24 28 32 36
Steamer Ream, Load 267 Cars
Hatches 4 8 12 16 20 24 28 32 36
Steamer Gerry
Hatches 4 8 12 16 20 24 28 32 36
20
10
0

Fig. 12.—Distribution of Cargo.

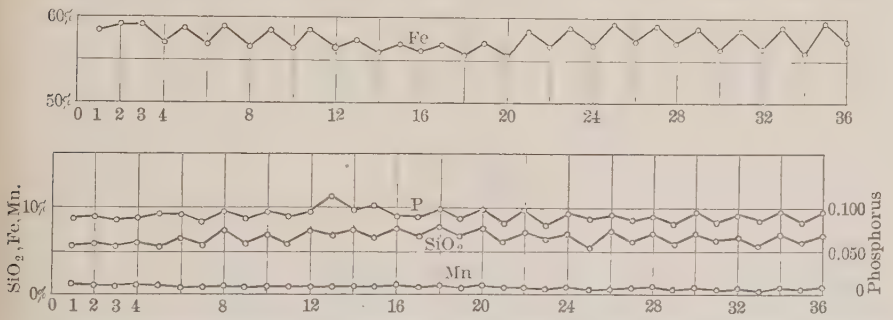
taken, might easily give a low result. It has been stated that from certain mines lump ore may be found which is of higher grade than the fine; I have no data indicating this. We may accordingly expect to find a difference between the finer center and the coarser borders of the pile, formed when a boat is being loaded.

Losses in Handling.—The loss of weight in transit due to drying is slight, but is more in loading and discharging. In the stock pile it may either dry out or take on moisture from rains, so that furnace weights afford no adequate check on the shipper's weights. One may note weigh scales forming part of the unloading rigs. These are not always in use. Often a part of the cargo is unloaded to cars to go to another works, part to the stock pile The cars are weighed, their aggregate subtracted from the reported weight of the cargo, and the remainder charged to the

stock pile. When one considers losses due to moisture and spilling this method seems inadequate for keeping a close record of quantities.

Time for Loading.—The cargo boats vary in capacity from 3,000 to 12,500 tons at a maximum draft of 20 ft. The typical lake ore carrier may be taken at 10,000 tons. When not overcrowded, and with the handling of ore and boat well attended to, a cargo is shot into a vessel in 5 hr., to which should be added 2 hr. for shifting to the desired pockets, and 2 hr. for waiting time. Another delay may result from a vessel arriving and finding another in advance of it, covering the pockets from which it has to draw.

Distribution of Cargo.—To oppose the buoyancy of the unloaded bow and stern, the boats are more heavily loaded at the end hatches, as graphically shown in Fig. 12, where the number of car loads per hatch is given. Now, in sampling, if equal weights of ore are taken at each hatch pro-



Railroad weight of cargo 11,793 tons. B/L weight, 11,678 tons. Average analysis: Fe, 58.94: P, 0.090; SiO_2, 6.48; Mn, 0.91; Moisture, 13.27 per cent.

FIG. 13.—CONTENTS OF CARGO, STEAMER "REAM."

portionate representation will not result. As far as my observation goes no such variation is allowed for in taking the sample at the lower-lake or receiving ports.

Cargo-Sampling Method.—The method recommended by the Chemists' Committee of the U. S. Steel Corporation for the sampling of a cargo, in a boat having hatches at 12-ft. centers, is, in any given hatch, to take a surface sample from the top of the heaped or coned ore, then, when the ore has been largely removed from a given hatch, a sample from the freshly exposed faces. The surface sample may amount to one-eighth, the face samples seven-eighths of the total. Now, were the last car loads shot into the vessel ore of high grade, the cargo sampler would get a high result; were they low, then samples would be depressed, even though the rest of the shipment were high grade.

Actual Results on a Cargo.—To arrive at the actual composition of a specific cargo I obtained from the train dispatcher at Hibbing, who assigns cars and makes up the blocks, the numbers of the cars and their composition, and from the dock superintendent, the pockets to which cars

are assigned. These cars constituted block 377, and were loaded into the boat "Ream." I also got from the dock superintendent the numbers of the pockets, and of the hatches of the vessel into which the pockets were discharged. In this way it has been possible to state with precision what has been the composition of the ore in each hatch, and that at the top surface of the coned-up ore. This is shown graphically in Fig. 13, in which the average analysis of the ore of each hatch is given.

In the following table are given the weights and contents of the cargo of the "Ream," 11,796 long tons, the gross weight being billed at 1 per cent. less than the railroad weight, or 11,678 tons.

TABLE I.—*Cargo of the "Ream"*

	Tons	Iron	Phosphorus	Silica	Manganese	Moisture
Hull Pit	5,529	60.74	0.080	4.56	0.71	14.34
Sellers Pit	1,524	58.26	0.091	8.90	0.42	10.19
Tener	272	54.07	0.085	6.86	2.26	11.13
Morris-Burt	714	57.42	0.083	6.28	1.32	12.20
Hartley	80	53.60	0.099	8.74	1.44	17.90
Burt	2,993	53.89	0.115	8.59	1.28	13.29
Glen	341	54.95	0.070	7.99	1.07	13.43
Clark	176	59 16	0.056	5.28	0.94	11.33
Winn-day	49	56.82	0.089	9.30	0.94	14.00
	11,678	58.94	0.090	6.48	0.91	13.27

Where, as in the shipper's sample, the actual weights and analyses that correspond are taken, the average of the cargo is 58.94 per cent. iron. On the other hand, assuming, as in cargo sampling, that an equal weight of ore is in each hatch, the resultant average will be 57.52 per cent. iron only. This will illustrate that in cargo sampling serious errors may result.

Complaint has been made by furnace superintendents that unexpected variations in cargoes, not shown by averages, have caused trouble in the working of blast furnaces. In this particular cargo the variations are not serious, as the figures given below indicate:

TABLE II.—*Variation of the Boat "Ream"*

	Iron	Phosphorus	Silica	Manganese	Moisture
(1) In ore delivered at the dock					
Maximum	60.74	0.115	8.90	2.26	17.90
Minimum	53.60	0.056	4.56	0.42	10.19
Average	58.94	0.090	6.48	0.91	13.27
(2) In ore delivered into the boat					
Maximum	59.50	0.114	7.80	——	——
Minimum	55.60	0.080	5.50	——	——
Average	57.40	0.092	6.60	——	——
(3) Surface sample of ore in boat					
Maximum	58.8	0.156	11.00	——	——
Minimum	51.9	0.067	3.90	——	——
Average	55.4	0.103	8.30	——	——

In this table we note, first, that as the result of spreading out of exceptional lots, and because of the consequent mixing, the ore has averaged better as it is shot into the hold. In (3) we see how a surface sample may vary from the average of the cargo, and also that if the ore from that point were introduced into the blast furnace it would cause more serious trouble than if the full depth in any hatch were used.

Stocking Ore at the Furnaces.—Ores of the same group are stocked

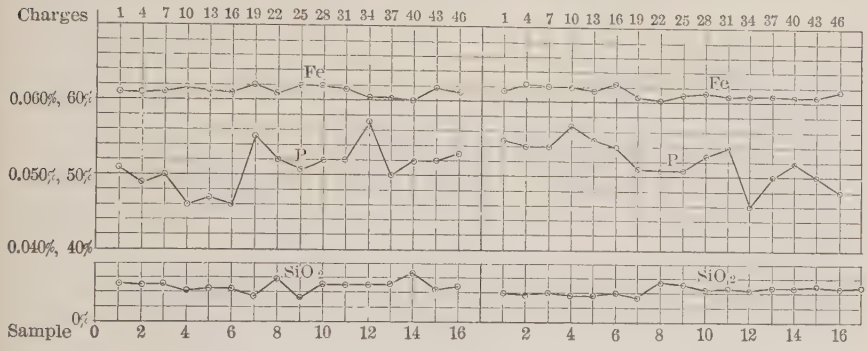


Fig. 14.—Analyses of Group 2 Ore from Charge Pockets.

together. If such ore were more carefully bedded, and for the furnace charges reclaimed from top to bottom of the pile, we would count on greater uniformity. If stocked haphazard from two or more cargoes, even when of the same group, the variations, when delivering from the receiving bins to the charge skips at the furnace, are shown in Fig. 14. The sample was taken as the ore was delivered into the skip at every skip load. It will be seen from the curves that at charge 34 in the first

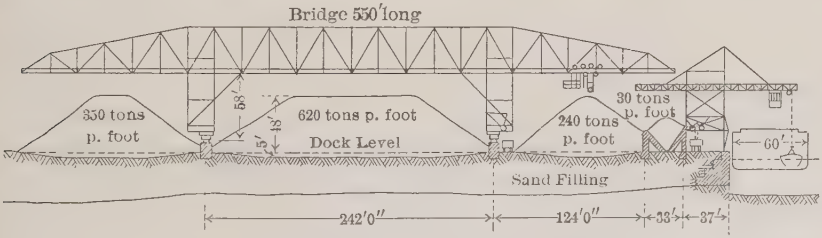


Fig. 15.—Distributing and Handling Bridge.

trial, and at charge 10 in the second, the phosphorus passed the limit which would permit the production of Bessemer pig. We may infer that the silica did not vary enough to cause a serious disturbance in blast-furnace working.

Stocking Iron Ore from the Boat.—Fig. 15 is an elevation of a Hoover & Mason unloading rig, of the stock piles made by it, of the concrete trough into which the ore from the vessel is delivered and from which it

is reclaimed to the stock pile; also a cross-section of the vessel being unloaded. Ore is taken from the boat in a 6-ton grab, which may deliver it either to a receiving hopper on the rig itself, or to the concrete trough behind it. Ore from any hatch is, at one setting of the unloading rig, delivered at one point at the trough. Here some segregation occurs, the lumps rolling to the foot of the pile. Thus we get a series of conical heaps of ore, their centers fine, lumpy where they overlap. The ore in the trough

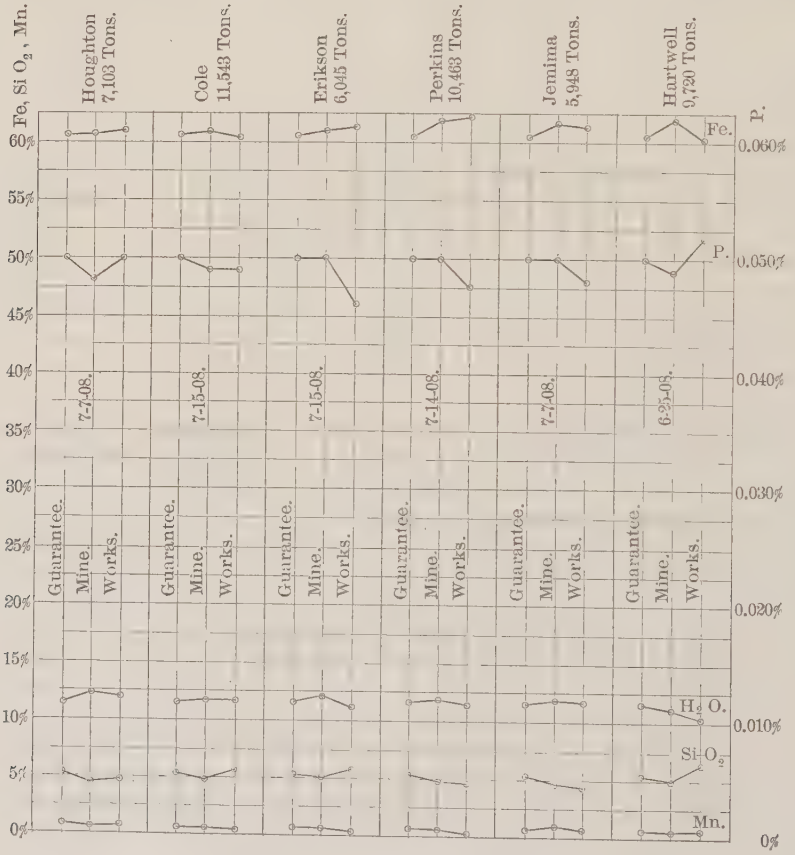


FIG. 16.—GUARANTEE, MINE, AND WORKS RESULTS ON CARGO SHIPMENTS.

is reclaimed and sent to the stock pile adjoining. Since, however, the two ends of the bridge may be independently moved to the extent of 30 or 40 ft. more at one end than at the other, it should be possible to so set it that coarse ore may be placed upon the fine, and thus some mixing be done. The grabs on the bridge are of 14 tons capacity. By carrying up the track walls of the bridge, it would appear that storage capacity might be increased, and the long slope be diminished. If, with this, attention should be given to carrying up the pile evenly and progressively

from the first, the ore bed would again be more uniform. Not sufficient attention has been given to this point and as long as the cargo was unloaded promptly it did not seem to matter how it was done. Ore naturally tends to segregate, and pains and study should be given to averaging it.

Cargoes, at the time of loading, are made up to a guaranteed composition. In Fig. 16 this composition is shown graphically, as well as the actual analyses of several cargoes as taken at the mine and at the receiving port during 1908 on Mesaba iron ores. As regards the mine and cargo samples there are plus and minus differences, indicating that the results are intended to be fair, though the cause of difference or the comparative accuracy is not indicated. My own opinion is that, as taken at present, the mine sample is the more reliable, though it could be improved. It has the advantage that numerous results are given, and the actual weight of the ore is used in computing the average.

A sample of the ore taken automatically, as it drops from a grab or from a Hulett machine, would give an aliquot portion of both coarse and fine. This sample could be kept in a pocket until taken away in a drop-bottom tram car. Such a car would deliver to the sampling mill, where the material would be automatically sampled by well-known methods. The system should be properly worked out so as not to interfere with the unloading. I compute that for a 1 per cent. sample, taken from the falling ore, it would cost 0.75c. per ton.

Of the present mehods of cargo sampling, that carried out by some of the independent chemists at Cleveland seems to me the best. They use a short-bladed scoop which takes a portion of uniform width. Wherever a lump of ore is encountered the sampler judges how much of it he is to take. The method of taking is by rounds, as described by Mr. Murray. The moisture sample is taken from the ore samples as collected in a can at each hatch. The contents of a can are emptied on the floor, the lumps broken, the whole mixed and returned to the can. Later a weighed portion of this is taken for drying at 212° F., and the moisture loss determined. The united ore samples of all the hatches are used for the average sample.

When it is considered that $50 to $75 is paid for the work of sampling and analyzing a cargo, the remuneration seems small for the care and responsibility involved; it amounts to 0.10c. to 0.15c. per ton. Compare this with the cost of sampling a precious-metal ore. Here the shipper cheerfully pays 50c. per ton if it is sampled at a customs sampler, 5c. to 10c. per ton if he engages a watcher to oversee the works sample. Nothing justifies spending only $50 to $100 to obtain results on material worth $40,000.

At the furnaces, sampling seems to be done only with the care needed for controlling blast-furnace operations, and to get results that will

check the mine. The blast-furnace superintendent seems to know quite well that the results of sampling are approximate only. I cannot see how he would feel like taking the responsibility of incurring expense, additional thought and pains to change the present system.

The U. S. Steel Chemists' Committee recommends a method of sampling called by them "grab" sampling; that is, taking a scoopful from the top surface of the ore in the grab as it is rising from the hatch. The method has much to commend it, as there is an equal representation of ore throughout the cargo; the chief objection to it in my mind is that the lumps would not be properly represented in the hasty taking.

Indeed, too much in regard to sampling is a matter of opinion and theory, and such conjectures could be set at rest only by sampling car loads, etc., in an absolute way, and by comparing these results with the present ones.

One may say then that sufficient attention has not been given to the sampling and stocking of ore, and that in consequence variations that occur here and there in a cargo, may appear at the surface in an aggravated way. This is the converse of the proposition that errors are disposed to balance one another, but it also logically leads to the admission that errors may be cumulative.

In view of the conditions of ore handling, I see no way for obtaining a uniform ore charge, unless each plant takes care of and beds its own ore. It has no right to complain of the shipper, who has only to guarantee the whole cargo.

A number of other points, not alluded to in this discussion, were developed as the result of my investigation, all of which, if put in practice, would conduce to reliability in sampling and to regularity of the ore charge, and consequently to regular running and to uniformity of the pig produced.

Notes on Conservation of Lake Superior Iron Ores[1]

BY C. K. LEITH, MADISON, WIS.

(New York Meeting, February, 1914)

THE questions as to what grades of ore it pays to conserve for the future, and the valuation of low-grade reserves, are becoming immediate and definite as applied to individual ore deposits, and seem to call for some sort of quantitative answer. Both opinion and practice involving these questions differ widely. The number of variables in this problem is so large that final solution will perhaps never be agreed upon, yet there seem to me to be certain controlling factors which can be placed in quantitative relation and serve to reduce the number of variables which it is necessary to consider in a given instance.

I assume that it will not pay either for the individual or for the State to conserve any ore when the present cost of conservation (by stock piling or otherwise), when compounded at 6 per cent. interest for the number of years which would elapse before this ore is used, will exceed the profit above current cost of extraction (not including interest) which the ore will yield when it is used. It may be urged that any loss to the individual may be made up in gain to the State. This might be true if there were any definite limit to the amount of iron ore, but when we remember that iron makes up 4 per cent. of all the rocks of the earth and the amount of rock carrying iron in excess of 20 per cent. is, for practical purposes, unlimited, it is apparent that posterity is assured of its iron ore. The changes will be mainly in methods and cost of extracting the ore. Therefore, any expenditure of money or effort now, which when compounded will exceed the worth of the effort or money needed in the future to utilize the iron, is a net loss both to the individual and to the community.

On the accompanying plate, Fig. 1, prepared by W. J. Mead, the amount of any given sum compounded at 6 per cent. interest for a given

[1] The subject of the economic possibilities of conservation is approached by the economists from a point of view which is perhaps new to some mining men. The reader may be interested in two short general papers on the subject, entitled:

Some Economic Aspects of Conservation, by Ralph Henry Hess, *Original Communications to the Eighth International Congress of Applied Chemistry*, vol. xxvi, p. 711 (1912).

The Economic Possibilities of Conservation, by L. C. Grey, *Quarterly Journal of Economics*, vol. xii (May, 1913).

period up to 40 years may be read. For instance, if it cost 10c. a ton to conserve ore, at the end of 20 years this amount will be 32c.

The present cost can be easily determined. Both the time at which an ore will be used and the future profit above mining costs are matters of speculation, but I believe it is possible to set certain limits for them.

Future Profits.—Just as ores now show the widest variety of availability, depending on chemical and mineralogical composition, texture, geologic and geographic position, cost of mining, etc., so the ores of lower grade which will be used in the future will show like variation in availability. The only factor common to all ores which can be used to summarize and compare the different ores is profit per ton. Profit determines whether a given ore can now be used and will determine this in the future. If, therefore, we can approximate the profit per ton above current cost of mining for any given ore, at any given time in the future, we can easily calculate how much, if anything, one can afford to spend now to save this ore. This profit can never be accurately predicted in advance, but I believe it may have much the same relation to profits on higher-grade ores as it has at present. If a 50 per cent. ore now shows a mining profit of $1.50 per ton, the lower grades as calculated from penalties will show profits of a fraction of this amount, ranging down to zero. In certain favored Lake Superior localities 38 to 40 per cent. low-phosphorus, high-silica ore can be mined now at a profit of 25c. or less. In less-favored localities, where now only the higher-grade ores are being mined, the time will come when the lower-grade ores will necessarily be mined; but when this time arrives, the profit per ton will probably be at least as low as the present profit for such low-grade ores mined from the most-favored localities and as low as the price we now figure from penalties, for the tonnage of the lower-grade ore available is so vast and so widely distributed that its extraction will be necessarily accomplished on a small margin of profit, like a manufacturing enterprise, such as, for instance, the utilization of clay and cement materials. The total cost to the consumer may be the same, but relatively more of the cost will go to fuel and to handling the larger quantities of material necessary to produce a given unit of finished product.

More specifically, I cannot see that there is any basis for hope that 40 per cent. Mesabi ores will be mined at any more profit per ton than the Marquette low-grade ores. When these 40 per cent. Mesabi ores come to be mined they will be in competition with enormous quantities of 40 per cent. ores elsewhere, and they will be mined at a very low margin.

Time.—For any given iron-ore district, the rate of exhaustion of the higher-grade reserves may be roughly figured and the time estimated in which the successively lower grades are likely to come into use. For instance, the average percentage of iron in the Lake Superior ores has been

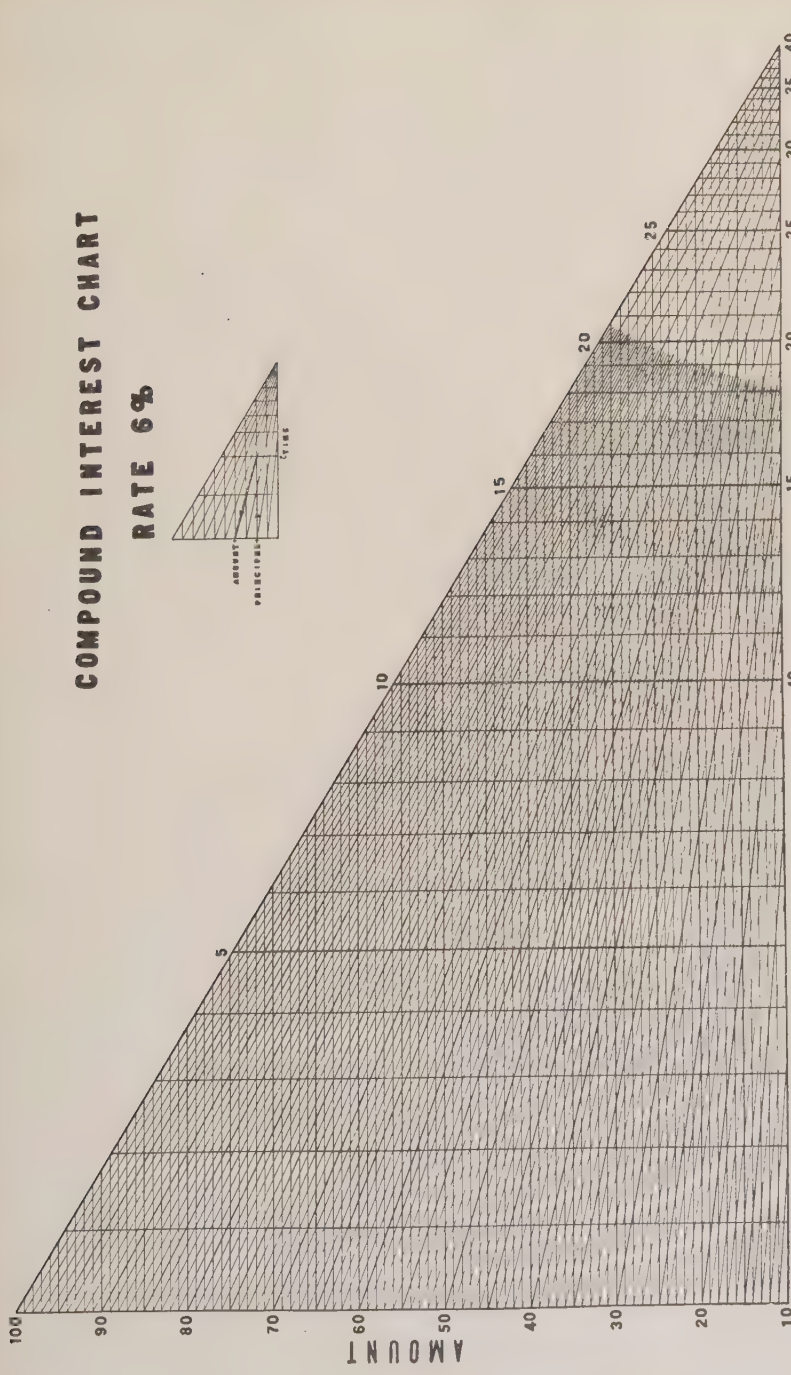


Fig. 1.—Chart for Determining Compound Interest at 6 Per Cent. for Any Ferod Between 1 and 40 Years.

showing during the past decade a drop of about a third of 1 per cent. per year. With a present average of about 49 per cent. natural, at the present rate of decrease of grade a 45 per cent. ore would come to be used in about 12 years. This time, of course, would not be uniform for all districts. Certain ores will be used long before others. This is a minimum time, because we must remember that as the grade falls the tonnage of available ores is enormously larger, which will slow up the rate of the decrease of grade. Also it is to be remembered that really only a small portion of Lake Superior is thoroughly explored. I suspect that, outside of the Mesabi, for a hundred years to come high-grade reserves will be added to more or less in proportion to the expenditure made in exploration. Finally, there is the probability of effective competition of foreign ores. I should say, roughly, that in any given district, one would need to be optimistic to predict large use of an average grade 5 per cent. lower than the present prevailing grade in a period shorter than 30 years.

Interpretation of Chart.—With these facts in mind, let us assume that it costs now 25c. per ton to conserve an ore which is 5 points lower than the present commercial grade. Let us also take an optimistic view of the situation and say that this lower grade when mined is going to yield a profit of 50c. a ton, which is perhaps as good as anything that could be figured out from the present profits on such ores, or from the calculation of penalties. It is apparent from the chart that such an ore must be used in 12 years unless there is to be a loss on the money invested in conserving the ore. If, on the other hand, it costs only 10c. per ton to conserve this ore, and the profit is assumed to be 50c. per ton in the future, it would be possible to hold the ore 28 years. If the profit per ton is assumed to be 25c. and the present cost of conserving it is 5c., it is possible to hold it for 28 years.

Without going into further illustrations, and without arguing for any specific figures of time and profit, it seems to me clear that there is a narrow limit to the sums which it is wise to spend to conserve low-grade ores. It is my opinion that 25c. per ton profit at a period over 30 years away may be the condition of mining the greater part of the ores 5 points lower than the present standard for any locality. It is apparent that it would pay to spend a very few cents a ton to conserve these ores. If this is true, the present value per ton in the ground of such low-grade ores is practically negligible.

In the above discussion royalty is included as a profit. The payment of royalty to the fee owner by the lease holder amounts to a division of profit. The profit of either party would, therefore, be lower than the figures I have given. It follows that royalties on low-grade ore will need to be pretty low.

My remarks refer principally to the direct use of raw ores. So far as beneficiation may come into use it will make low-grade ores more

immediately available, and therefore increase their value, but I take it that deducting interest, depreciation and profit on the beneficiating plants themselves will leave such small profit on the low-grade ores themselves as not seriously to modify the above principles.

The above suggestions are offered tentatively in the hope that they may bring out some discussion.

Discussion

Alfred C. Lane, Tufts College, Mass.: I entirely agree with the conclusions of Mr. Leith's paper. The value of reserves which are not to be used for many years is much smaller than is sometimes realized. It may also be said that in assuming 25c. a ton profit and royalty at distant dates, Mr. Leith is granting all that can probably be asked, for I think there is a widespread tendency toward reduction of profits. Possibly Mr. Leith has some figures showing what the reduction of the mine royalties given at different dates in the past has been. With the lowering of the grade of iron ore, it is likely that the royalty and profit will also lower per ton and the tendency of the times to tax land values and the unearned increment must not be forgotten.

Notes on an Iron-Ore Deposit near Hong-Kong, China

BY C. M. WELD, NEW YORK, N. Y.

(New York Meeting, February, 1914)

THE southeastern coast of China, from Ning-Po to Macao, represents an element in the continental mass of Asia which has at practically all times in the remote past exhibited a tendency to rise rather than to subside.[1] This strip of land, 1,000 or more miles long by perhaps 150 miles wide at its widest part, has in fact stood above sea level since at least early Cambrian times, while the great basins to the north and west were receiving their massive accumulations of Sinian (Cambro-Ordovician), Carboniferous, and later deposits.

We find, therefore, a coastal fringe of metamorphic and igneous rocks bordering the huge expanses of sedimentary rocks which occupy the interior. Hong-Kong is situated toward the southwestern extremity of this fringe.

Hong-Kong is an island of approximately 30 sq. miles, on the north shore of which is situated the city of Victoria with its 325,000 inhabitants. This island was ceded to Great Britain in 1842. In 1860 the peninsula of Kowloon (about 5 sq. miles), being that part of the mainland lying directly across from the city of Victoria, was added to Hong-Kong. Thirty-eight years later, in 1898, a stretch of territory, including the mainland behind Kowloon and numerous adjacent islands, was leased to Great Britain for 99 years. This last addition is known as the "New Territory of Hong-Kong" and comprises 376 sq. miles in area.

A map of Hong-Kong, Kowloon, and the "New Territory" is shown in Fig. 1. My studies of this total area of over 400 sq. miles were devoted primarily to an investigation of the mineral resources. Systematic mapping of the immensely complex exposures of rocks was not possible in the time at my disposal. However, observations in this direction warrant a few generalizations.

The area in question is largely occupied by granite, for the most part rather coarsely crystalline, with quartz, potash-feldspar, hornblende, and subordinate quantities of mica as its chief constituents. Scattered throughout the granite are irregular masses of highly metamorphosed

[1] Bailey Willis: *Research in China*, vol. ii, chap. viii (Carnegie Institution, Washington, 1907).

sediments, in fragments ranging in size from a few cubic feet up to the immense quartzite peaks of Ma-On-Shan mountain. At and near contacts the granite exhibits fine-grained and porphyritic facies. In plan the area may be conceived of as a highly irregular mosaic, with baked sedimentary fragments of every size and shape set rather sparsely and absolutely at random in a matrix of granite.

FIG. 1.—MAP OF HONG-KONG NEW TERRITORY, SHOWING IRON-ORE DEPOSITS.

Among the metamorphosed inclusions the most abundant are quartzites. Next come graphitic schists and clay slates. These are particularly abundant toward the northwest, finally coming in in great force and to the total exclusion of the granite along the northwest frontier, where they strike northeast to southwest and dip to the northwest. Occasional evidences of impure limy or magnesian carbonate rocks were also observed.

Along the eastern shore of the "New Territory," from Grass Island to Fo-Tau-Mun, large masses of basalt were seen, exhibiting marked

columnar structure. Von Richthofen[2] refers to the presence of "quartzose porphyry and its tufas" on the island of Hong-Kong; these were not observed by me.

We have then an area occupied chiefly by an intrusive granite, carrying immense numbers of baked sedimentary inclusions of every size and shape. Along the northwest border of this area the granite gives way to schists and slates, dipping to the northwest. Along the eastern and southeastern edge are masses of basalt and quartz-porphyry.

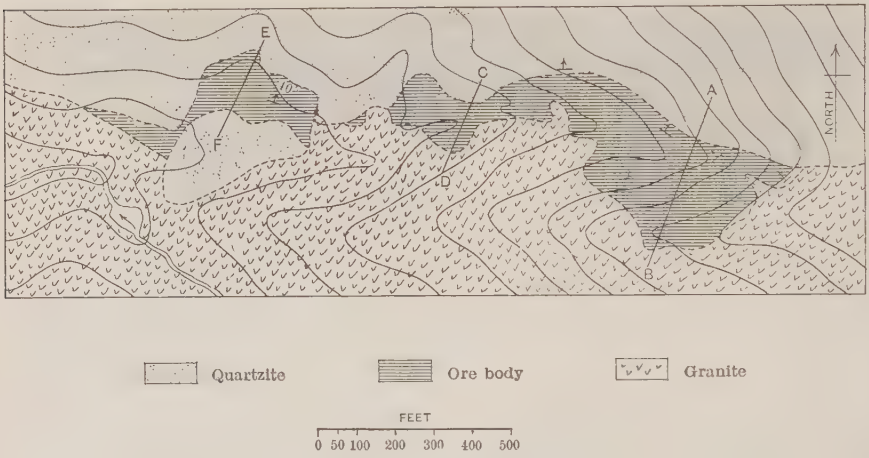


FIG. 2.—GEOLOGICAL MAP OF THE MA-ON-SHAN IRON-ORE BODY.
Contour Interval, 40 ft.

The geologic history of this area may be readily interpreted. Sediments, probably of pre-Cambrian age, including sandstones, shales, and a few impure limestones or dolomites, were invaded at depth and shot to pieces by immense masses of granite. The northwest border of the intrusion appears to have more or less coincided with the present northwest frontier of the "New Territory." (The frontier line was in fact established for strategic military purposes so as to include the coastal mountain ranges, and divides these from the lower-lying areas of the interior.)

Vertically upward movements followed upon the period of intrusion and have continued to persist since early Cambrian times, maintaining the "New Territory" continuously above sea level, and bringing about the ultimate exposure through denudation of what is probably not far from the summit of the granite batholith, with its included fragments. At some period later than that of the granitic intrusion, renewed volcanic activity gave rise to the basalts and porphyries.

[2] *Proceedings of the American Academy of Arts and Sciences*, vol. viii, p. 117 (1869).

Topographically the "New Territory" presents a land surface only moderately advanced in its erosional cycle. A few broad valleys have been carved out and all traces of a possible ancient peneplain have been removed. For the most part, however, the land is mountainous, with peaks rising as much as 3,000 ft. above sea level, frequently exhibiting steep slopes and torrential streams. Erosion processes have still much work to perform to balance the effects of the last period of elevation.

In the meanwhile we find evidences of a present reversal of the usual order of things. A glance at the map, Fig. 1, reveals numerous deep

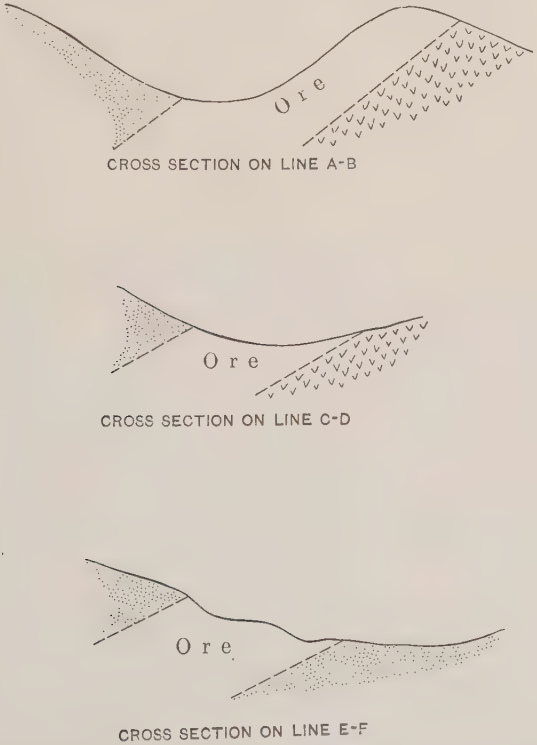


FIG. 3.—CROSS-SECTIONS, LOOKING EAST, ON THE LINES A–B, ETC., OF FIG. 2.

indentations and peaked islands, with deep-water harbors and channels close to shore. In short, the "New Territory" is clearly to-day a good example of "drowned" topography. Thus the latest chapter in our history points to a period of vertical subsidence, following upon the tendency to rise which has persisted through so many geological eras.

It is not surprising that an area with a history such as the above should show abundant and varied evidences of mineralization. Gold and silver, lead, zinc, copper, and tin have all been discovered at various localities. Iron-ore deposits have also been recorded, including secondary

surficial deposits, occasional small veins, and contact deposits. One at least of this last type possesses economic interest and it is my purpose to present a description of it.

The Ma-On-Shan iron-ore body lies somewhat less than a mile southwest of Ma-On-Shan peak. Its position is marked on Fig. 1. The distance in a direct line to Tide cove is about $1\frac{1}{2}$ miles, and not more than $2\frac{1}{2}$ miles to the navigable waters of Tolo harbor.

This ore body has been explored to some extent. It is first seen about $\frac{1}{2}$ mile southeast from the village of Ma-On-Shan and extends in an easterly direction up a small creek for a total distance of nearly 2,000 ft. Its western extremity lies 800 ft. above sea level and its eastern extremity 1,200 ft. Ma-On-Shan peak rises to an elevation of 2,260 ft.

Fig. 2 is a geological map of the ore deposit. Three types of rocks are distinguished, these being, in general, quartzite on the north, ore

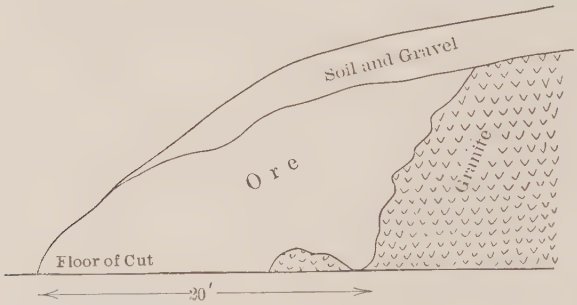


FIG. 4.—SKETCH OF EAST WALL OF SOUTHWARD-BEARING CUT, SHOWING CONTACT BETWEEN ORE AND GRANITE.

and its associate in the center, and granite on the south. The ore bed and quartzite dip toward the north at from 30° to 60° from the horizontal. At one point a considerable area of quartzite intervenes between the ore and the granite, and evidences of the same relations are found toward the east, where a southward-bearing tunnel started well within the ore area encountered a foot wall of quartzite instead of granite. A drill hole also found quartzite underlying the ore. At most points where the contact is observable, however, the ore lies directly upon granite.

It is apparent that the sedimentary rocks, including sandstone and an intercalated bed of what is assumed to have been originally an impure dolomite (to-day represented by the ore and its associate—this point will be discussed more fully below), have been intruded by great masses of granite. The contact shown on the geological map (Fig. 2) is a small portion only of the southern edge of an immense fragment of quartzite, constituting Ma-On-Shan mountain, and entirely surrounded by granite.

This fragment possibly represents an uneroded portion of the original cover resting upon the summit of the granite batholith. At the point where the ore occurs the granite penetrated but did not entirely remove the dolomite bed. Thus a few shreds of dolomite were here left clinging to the edge of the sandstone, partly protected by remnants of the underlying sandstone. Both to the east and to the west the dolomite bed either did not originally exist or else was torn away and removed by the granite. At least, no trace of it appears to-day and the quartzite is seen in direct contact with the granite.

South of Starling inlet, 8½ miles due north from Ma-On-Shan, a lenticular bed of magnesian or limy shale was discovered, of limited extent and grading on its strike into sandy shale. It is believed that this may be in some degree a counterpart of the original rock of the Ma-On-Shan ore body.

Three more or less idealized cross-sections of the Ma-On-Shan ore body are shown in Fig. 3. The sedimentary beds are seen dipping to the north. In one case quartzite is shown underlying as well as overlying the ore body. Just what the altitude of the granite contact may be is problematical. At the few exposures where it was possible to observe conditions the contact line was sharp but highly irregular, with a general dip toward the north. Fig. 4 is a sketch of the east wall of a small southward-bearing cut which crossed the contact.

That the sedimentary rocks have been dislocated is abundantly shown by the geological plan, Fig. 2. At least three considerable faults are seen, trending north-northeast to south-southwest, with the down-throw on the northwest side. The westernmost fault was clearly revealed by a tunnel which started in the overlying quartzite and bore eastward through the fault zone into the ore body. These faults do not cut the granite. They either antedated the intrusion or possibly were due, in part at least, to the shock of this event.

The granite is similar to that found throughout the Hong-Kong Territory. Its essential constituents are quartz and potash-feldspar, with subordinate quantities of hornblende and mica, chiefly muscovite. No microscopic examinations were made. At or near the contact it is fine-grained to (occasionally) porphyritic, in the latter case exhibiting at times conspicuous twinned orthoclase phenocrysts. Away from the contacts the texture is coarser. In color it is usually light gray, but more rarely pink, due to the presence of pink feldspar.

The quartzite lying north of the ore body is rather sandy in texture, generally altered and stained; no fresh exposures were observable adjacent to the ore body. The underlying quartzite is a dense, exceedingly hard white rock, with few or no minerals other than quartz visible to the naked eye. At one point scant veinlets of molybdenite were found.

The ore-bearing bed furnishes a most interesting study. I have referred to this as having probably originally been a bed of impure dolomite; a suggestion which carries with it a definite hypothesis regarding the genesis of the ore body, and which therefore requires development. As already stated, it dips toward the north at from 30° to 60° from the horizontal. It is about 100 ft. thick where its full thickness can be measured between quartzite walls, but not so thick where it lies upon granite.

It is not a solid mass of iron ore, but is broadly divisible into two members: namely, pure magnetite, and what was termed provisionally in the field "greenstone," but which closely resembles the "skarn" of the iron-ore belt of central Sweden. I shall therefore call it skarn in the following. This is a deep green rock, breaking into angular fragments where weathered, generally aphanitic, but from time to time presenting to the naked eye one or more of the following minerals: magnetite, quartz, garnet (frequently abundant), pyroxene, amphibole, mica, chlorite, epidote, serpentine.

The relations of the magnetite and the skarn are highly irregular. As a rule, it may be said that the magnetite occurs abundantly disseminated throughout the skarn, in bodies ranging from isolated minute crystals up to pure masses comprising thousands of tons. In its pure, massive form the magnetite is generally rather soft and granular. There are no well-defined boundaries between the magnetite masses and the skarn, the one grading imperceptibly into the other. The skarn frequently carries sufficiently high proportions of magnetite to constitute a commercial iron ore in itself.

In the following table I present five analyses of Ma-On-Shan ores and skarn. Nos. I and II were taken from the richer portions of the ore body, while No. III represents magnetic concentrates. In this case 22 samples of lean ores and skarn, taken from all portions of the deposit, were treated in the laboratory with a horseshoe magnet and the concentrates thus obtained were sent up for analysis. No. IV was an average sample of ore, and No. V was lean ore or rich skarn. Analyses I and V were made at the laboratory of the Langloan Iron & Chemical Co., Coatbridge, England; analysis II at the Gwendoline mine, Korea; analysis III by Riley & Co., Bombay, India; and analysis IV by Pearson Claudet, London, England.

On viewing these analyses we note that not all the iron occurs as magnetite, there being a considerable proportion of hematite, particularly in analysis III.

A most striking feature of the analysis is the great amount of magnesia, especially as compared with the almost insignificant amount of lime. The five results given in the table do not, however, represent the invariable ratio of these two elements. Thus eight other samples, all

of skarn, analyzed in a private laboratory in Hong-Kong, yielded an average of 6.05 per cent. CaO and 5.58 per cent. MgO, with considerable amounts of "insoluble matter."

	I	II	III	IV	V
Fe_2O_3	70.32	65.25	75.36	62.24	38.40
FeO	22.53	22.26	11.37	15.75	15.84
MnO	1.48	.....	0.70	1.03	0.74
SiO_2 }	1.20	3.99	5.50	6.04	12.70
Al_2O_3 }		.....	4.46	1.80	1.21
CaO	0.60	.....	0.73	0.56	0.40
MgO	3.64	Trace	2.29	11.53	26.24
S	0.110	0.27	0.082	0.09	0.192
P_2O_5	0.009	0.044	Trace	.....	0.028
As	0.008	.....	Nil	.....	0.005
CO_2	.....	.....	Nil	0.80	1.50
Water	.....	7.20	Trace	.....	
Moisture	.....	.....	.....	.....	0.10
Loss	.....	.....	.....	0.16	.....
Total	99.897	99.014	100.492	100.00	97.355
Fe	66.75	63.00	61.59	55.82	39.20
P	0.004	0.019	Trace	.....	0.012

Four specimens were submitted to microscopic examination, as follows: (*a*) and (*b*), typical skarn with angular jointing; (*c*), skarn with iron ores and garnet; (*d*), skarn, iron ores more prominent and showing quartz, feldspar, and ferro-magnesian minerals. The following descriptions of the micro-slides were returned, giving the minerals observed in the order of their quantitative importance.

(*a*)

1. Uralite.
2. Colorless pyroxene?
3. Magnetite.
4. Periclase?

(*b*)

First Slide

1. Uralite.
2. Colorless pyroxene?
3. Magnetite.
4. Periclase?

Second Slide

1. Uralite.
2. Magnetite.
3. Periclase?
4. Pyroxene?

(*c*)

First Slide

1. Colorless twinned pyroxene.
2. Magnetite.
3. Secondary aggregate formed from 1.

Second Slide

1. Pyroxene.
2. Magnetite.
3. Secondary serpentine.

(*d*)

1. Pyroxene?
2. Magnetite.

The mineral determined as periclase (described as a colorless isotropic mineral with a high refractive index) was so designated only with hesitation. However, the excess amount of magnesia shown by the analyses lends generous support to this determination, leading to the supposition that when the rock was developed into its present state a part of the magnesia combined with the silica, lime, and alumina to form pyroxene, while the excess remained as free magnesia or periclase.

Taking the results of the micro-slides broadly, in conjunction with the analyses, it would appear that the typical unaltered aphanitic portions of the skarn consist essentially of lime-magnesia-pyroxene, magnetite, and periclase. The more coarsely crystalline portions reveal, in addition to magnetite and not readily distinguished ferro-magnesian minerals, quartz, frequently abundant garnet, mica, chlorite, epidote, and serpentine.

Without the field evidence the magnetite-skarn body might be adjudged a magmatic differentiation out of an ultra-basic rock. As it is, it is believed to be of contact-metamorphic origin. It seems probable that this ore-bearing bed, lying partly intercalated with quartzites and in contact with an intrusive batholith of granite, has been developed out of an originally non-ferruginous but easily vulnerable rock through the selective impregnation of mineral-bearing emanations from the granite magma.

Examples of contact metamorphism where limestone beds have been locally wholly replaced by iron ores, associated with a variety of minerals generally similar to those found in the Ma-On-Shan ore body, are well authenticated and numerous. Perhaps the only striking difference between these authenticated occurrences and Ma-On-Shan is the extraordinary abundance of magnesia in the latter. This we may account for by the obvious suggestion that the original rock was one rich in magnesia—namely, a dolomite.

The high-magnesia characteristic is not entirely unique, as may be seen by comparison with the published analyses of iron ores from the Tilly Foster and Mahopac mines of Putnam county, N. Y.;[3] in that case, however, the magnesia is identified as present in the mineral chondrodite. Koeberlin[4] attributes a contact-metamorphic origin to these ore bodies. Sjögren[5] describes a similar occurrence and association at Nordmarken, Sweden.

The fact that the Ma-On-Shan deposit, as far as observable, is in the form of a bed does not preclude a contact-metamorphic origin for its ores. It would be but natural that where an intrusive has cut a series of beds of varying composition and texture, a selective attack, according to the vulnerability of the different materials presented to its field of action, has taken place.[6] In this case the sandstone has suffered little

[3] A. F. Wendt: *Trans.*, xiii, 481 (1884–85).

[4] *Economic Geology*, vol. iv, No. 8, p. 749 (Dec., 1909).

[5] *Trans.*, xxxviii, 796 (1907).

[6] I may cite as a striking example of such a selective attack the magnetite ore beds near Dillsburg, Pa., described by Spencer (*Bulletin No.* 359, *U. S. Geological Survey*, pp. 74 to 96 [1908]) and Harder (*Economic Geology*, vol. v, No. 7, p. 599 [Oct.–Nov., 1910]). The Mahopac ore beds above referred to are also an excellent example.

beyond an intense baking, possibly also silicification. The dolomite bed, on the other hand, has been thoroughly saturated by the magmatic emanations. Carbonic acid has been practically wholly expelled (see analyses IV and V), as also probably a portion of the lime. Magnesia appears to have been more resistant and has largely remained; in part as periclase, in part combined with other elements to form new minerals. Large amounts of iron have been added and possibly small amounts of silica. In short, the dolomite bed has been converted in the vicinity of its contact with the granite into a magnetite-skarn bed, the latter following the lines and structural position of the former. If the ore bed could be followed to a sufficient distance away from the granite contact, it would probably be found in its original condition of impure dolomite.

The fact that the possibly pre-granitic faults cut the ore body does not indicate that the ore body likewise was pre-granitic. The dolomite bed suffered this faulting, and when replaced by ores and contact minerals these latter naturally followed out all of its structures, completely filling the spaces formerly occupied by it between the inclosing walls of impregnable sandstone.

I have indicated on Fig. 1 five other localities where iron-ore bodies similar to Ma-On-Shan but of smaller dimensions have been found. I have also shown three points where veins of magnetic iron lying wholly within granite areas have been discovered. These latter undoubtedly owe their metal to the same sources as the contact-metamorphic deposits, the veins being shrinkage fissures due to the cooling of the granite and filled by mineral-bearing solutions rising out of the still viscid interior. The presence of these fissure veins of magnetic iron further supports the hypothesis that the iron ores of Ma-On-Shan were derived from the granite and were not indigenous to the intruded sediments.

I conclude with the belief that there is sufficient evidence for assigning this interesting Chinese iron-ore deposit to the now well established contact-metamorphic or, as more narrowly defined by Leith,[7] "pegmatite" class of ore deposits.

[7] *Economic Geology*, vol. iii, No. 4, p. 277 (June–July, 1908).

The Plant of the Duplex Process for Making Steel

BY J. K. FURST,* NEW CASTLE, PA.

(Pittsburgh Meeting, October, 1914)

THE reasons for manufacturing steel by the duplex process are, briefly: saving of time; increasing output for capital invested; and avoiding the difficulty sometimes experienced in obtaining scrap. On account of the latter reason, some plants duplex only at such times as the price and quantity of scrap in the open market warrant. At some plants the use of the duplex process enables the blast-furnace practice to be simplified, because the furnaces run more smoothly and produce more iron when not held to a strict specification as to the silicon and phosphorus therein.

The duplex process was practiced in Witkowitz as far back as 1878, but it was left to the steel makers of this country to develop it along practical, commercial lines. The American companies manufacturing steel by the duplex process at the present time are: Tennessee Coal, Iron & Railroad Co., Maryland Steel Co., Bethlehem Steel Co., Pennsylvania Steel Co., Jones & Laughlin Steel Co., Colorado Fuel & Iron Co., Lackawanna Steel Co., Dominion Iron & Steel Co. of Canada.

All of these companies, except the Dominion Iron & Steel Co., combine the acid Bessemer with the basic open-hearth process, while the latter combines the basic Bessemer with the basic open hearth.

The Tennessee Coal, Iron & Railroad Co.'s Plant

The installation of a converter of 15 tons capacity at the Ensley plant of the Tennessee Coal, Iron & Railroad Co. in 1904 marked the introduction into this country of the duplex system. The experience gained in the employment of the Bessemer converter had demonstrated that there was a shortening of the time required in the open-hearth furnace and a reduction of the outlay for fuel and lime, thus affording substantial evidence of the economies resulting from this method.

The scarcity of steel scrap in the Birmingham district; its dependence on its own resources for pig iron; its lower cost of fuel; and the lower lime cost with the use of the direct metal in the converter, were the factors

* Chief Engineer, Pennsylvania Engineering Works.

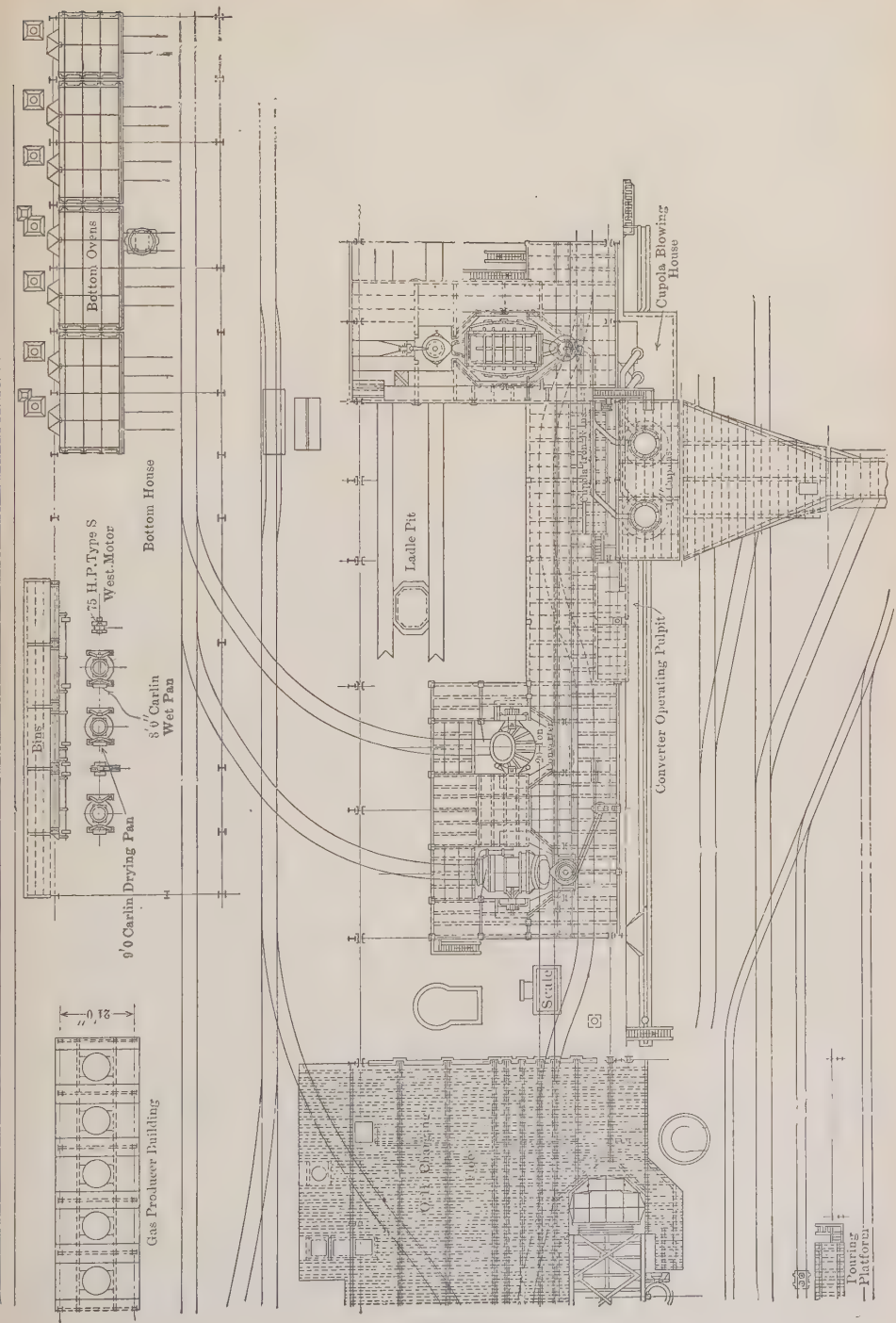


FIG. 1A.—PLAN OF THE DUPLEX PLANT OF THE TENNESSEE COAL, IRON & RAILROAD CO.

which brought about the construction of the second duplexing plant, which went into operation in the fall of 1907 and has been operated continuously ever since. At the time the original 15-ton converter was installed at the Tennessee Coal, Iron & Railroad Co.'s works, the steel-making plant consisted of six blast furnaces, one 250-ton primary regenerative furnace, and eleven 50-ton open-hearth furnaces, of which one was stationary and ten were tilting. The original converter was erected between the 250-ton regenerative furnace and the open-hearth plant, and was served by an electric traveling crane. It was operated by one 75-h.p. electric motor through worm, pinion, and gear, and was the first converter in this country to be operated electrically.

The second plant built by this company marked the beginning of duplexing in this country on a large scale. All the details of this plant were worked out so carefully that practically no changes or improvements have been found necessary, or even desirable, since then. As shown in Fig. 1, this plant consists of six modern blast furnaces and two batteries of basic open-hearth furnaces, each consisting of four 100-ton hydraulically tilting furnaces, having hearths 15 ft. wide by 44 ft. 2 in. long. Between the two batteries of open-hearth furnaces, and in line with them, is located the Bessemer building. The converters and the converter-building floor are served by a 100-ton traveling crane. This building houses the two 20-ton converters; one 250-ton and one 600-ton hot-metal mixer; and two 10-ton cupolas for melting scrap, located directly back of the pouring-end of the mixers, with the necessary equipment of troughs so that they can be tapped into the converter-charging ladle which plies between the mixers and converters.

The iron comes from the blast furnaces in 25-ton hot-metal cars, and is poured into the mixers by means of hydraulic lifts. The iron is poured from the mixers into the converter-charging ladle, which travels over a charging floor at an elevation of 14 ft. 7 in. above the open-hearth charging floor. The molten iron is charged into the converter by means of a hydraulic post crane. After it has been blown, it is discharged into a 20-ton ladle car and carried directly to the open-hearth furnace. The bottom house, situated directly opposite the converters, contains seven bottom-drying ovens, crusher, wet and dry pans, and the necessary bins, etc., for making up the bottoms. After the bottoms have been made up here and dried, they are placed on the hydraulic jack cars, of which there are two, and transported to the converters as required.

The distinctive feature of this plant is that the converters are located directly between, and in line with, the two batteries of open-hearth furnaces, thus minimizing the haul of the blown metal. The track over which the blown metal is conveyed to the open-hearth furnaces is located in the open-hearth building, adjacent to the furnaces and between the charging-box track and the furnaces. The ladles are drawn up in front

of the furnace doors and the contents poured directly from the ladle, without removing it from the car, into the charging spout. The ladle is tilted by an overhead traveling crane.

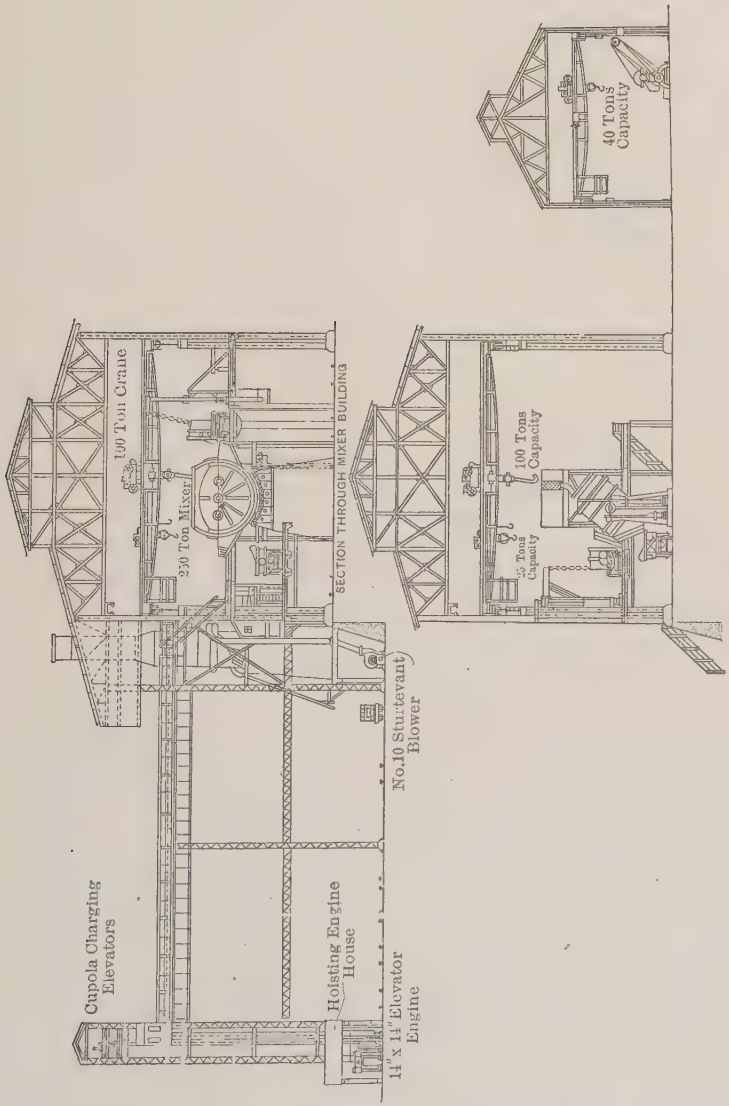


FIG. 1B.—SECTIONS THROUGH PLANT OF TENNESSEE COAL, IRON & RAILROAD CO.

The vessels, as shown in Fig. 2, are swung on well-designed, heavy, cast-iron column stands and are 40 ft. center to center. Instead of following the usual practice of riveting the trunnions directly to the shell, they are secured to an annular supporting ring, the vessel being secured

in place by means of lugs and keys. This supporting ring not only serves to prevent the wearing of the sides of the vessel in the rivet holes which have become distorted by the constant reversal of stresses on account of turning the vessel from time to time, and which has been a source of great inconvenience in the past at some of the works manufacturing Bessemer

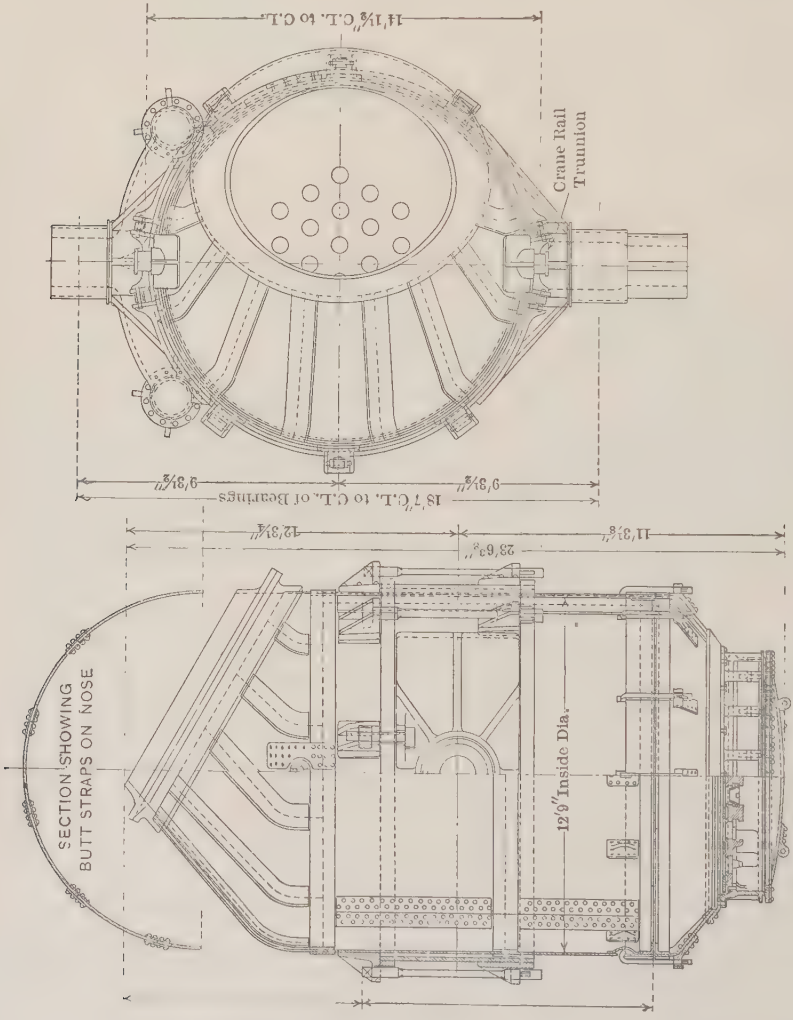


Fig. 2.—20-ton Converter, Tennessee Coal, Iron & Railroad Co.

steel, but also facilitates the removal of the vessel for relining and repairs. A spare vessel, lined and ready for service, is kept standing convenient to the vessels in operation. The tuyère plate has 26 tuyère openings. The vessels are operated hydraulically by two vertical racks working on one pinion for each vessel. Blast is supplied to the converter from an

Allis horizontal, cross compound, Corliss blowing engine (46 by 88 by 84 in. and 60 by 84 in.) through a 30-in. diameter air line to the 24-in. air valves beneath the pulpit, thence through an 18-in. air line to the blast trunnion of each vessel. The blast line, just before connecting with the trunnion sleeve, is provided with a gas-escape valve, which allows any back flow of gas to escape to the atmosphere rather than into the main air line. The blast elbow and wind box are of the usual design. The unusual size of the converters was a matter that received careful consideration at the outset by the Tennessee Coal, Iron & Railroad Co.'s engineers, and the 20-ton vessel was adopted, since it afforded economies in operation, by decreasing the refractory cost and reducing the expense of blowing.

While it is not the intention of this article to go into either the chemistry or the metallurgy of the duplex process, a brief outline of the method of operation of some of the followers of the process may prove of interest: The methods employed at this plant of bringing the iron from the furnace, charging the mixer, charging the converter, and pouring from the converter, have been previously described. The hot metal coming from the blast furnaces to the mixer is of the following analysis: Silicon, 0.80 to 1.25; phosphorus, 0.9 to 1.0; manganese, 0.3 to 0.4 per cent. In the converter, all of the metal is desiliconized and partly decarbonized. Four ladles of blown metal are required for each open-hearth heat. Ordinarily the first two are blown soft, analyzing: carbon, 0.1; phosphorus, 0.7 to 1.0; manganese, 0.08 per cent.; and the second two ladles are partly decarbonized, the percentage of carbon blown out depending on the amount of scrap charged in the open-hearth furnace. Before the blown metal is poured into the open-hearth furnaces, burnt lime, iron oxide (the latter in the form of scale or ore), and about 15 per cent. of scrap are charged; then the two ladles blown soft are poured in, and lastly the two ladles partly decarbonized, containing 2 per cent. carbon, or slightly more, are added. When the second two ladles are poured into the open-hearth furnace a violent reaction takes place, which transfers practically all of the phosphorus into the slag. It is here that the advantage of the tilting type of open-hearth furnace, over the stationary type, comes in, as at this juncture the furnace is tilted back, allowing the excess slag to run out of the doors into the slag cars beneath the furnace. Recarbonizing of the steel at the end of a heat is not often found necessary, the practice making it possible to catch the carbon on the way down. Ferro-manganese is added for the manganese. It took from 1 to $1\frac{1}{2}$ hr. to charge the blown metal into the furnace, and about 1 hr. to finish the heat. Records kept show a variation of from 4 to 8 hr. per heat. The time required for blowing the heats in the converters varies, depending on the silicon content, the blast pressure, and other factors. Desiliconizing takes from 2 to 10 min., and decarbonizing from 12 to 20 min.

The Dominion Iron & Steel Co.'s Plant

At about the same time the Tennessee Coal, Iron & Railroad Co., situated at the southern American boundary of the steel industry, was considering the question of duplexing on a large scale, the Dominion Iron & Steel Co., situated at the extreme northern boundary of the steel industry, concluded to install a duplexing plant. The Nova Scotian ores were similar in many respects, but had a higher phosphorus content, and the chief reasons why duplexing was considered advantageous were the long period of time otherwise required in the open hearth, and the difficulty of operating the blast furnaces under a strict specification as to silicon and phosphorus. Consequently, in July, 1906, they let the contract for two 15-ton Bessemer converters, with the necessary building and equipment. Construction started in December of the same year and the first heat was blown in May, 1907. The vessels were operated for some time with an acid Bessemer lining with very satisfactory results, which, however, seemed short of the maximum possibilities. It was thought that, by operating with a basic lining, better results might be achieved. This was at the time considered somewhat experimental in view of the fact that in Europe, where the basic Bessemer practice was commonly pursued, it was considered that at least 1.75 to 2.25 per cent. phosphorus was necessary. However, the necessary alterations in the bottom house were made, the mica schist lining of the vessels was removed and one of stamped dolomite and tar substituted, and the basic Bessemer process was begun. This method has been in use ever since.

In the year 1910 a third vessel, of the same size and design, was added to the original plant. At this time, the steel-manufacturing department consisted of four blast furnaces, one 300-ton hot-metal mixer, three 15-ton Bessemer converters, two in actual service and one spare, and ten 50-ton basic open-hearth furnaces of the Campbell tilting type. Of the ten open-hearth furnaces, nine are operated according to strict open-hearth practice, furnace No. 1 alone being used in conjunction with the duplex process. Also, there were under construction at this time two blast furnaces and two 500-ton mixers. The mixer building and the converter building are situated between the blast furnaces and the open-hearth plant. The vessels are served by a traveling crane, which operates throughout the full length of the building. The three vessels are arranged in line in the Bessemer building, 36 ft. centers. The vessels are of the eccentric type, having a shell diameter of 10 ft. 9 in., with trunnions riveted directly to the shell. The wind box, blast elbow, and trunnions are of the usual design; but the tuyère plate, instead of being of the usual type with 6-in. openings in which refractory tuyères are placed, is a solid plate provided with 73 $\frac{3}{4}$-in. tuyère openings. The vessel is electrically operated, the power being transmitted through worm, pinion, and gear

directly to the trunnion, which gives ample power at all times, although when the vessel is in a horizontal position with bottom off, it requires about all of the power to bring it to the vertical position. The blast for blowing the vessel is furnished by the blast-furnace blowing engines through a 36-in. diameter main, 1,365 ft. long. We mention this length on account of the unusual distance over which the blast is carried. The 18-in. branch blast mains on each vessel are provided with air-relief valves. The blast is delivered to the vessels at a pressure of 18 to 20 lb.

At this plant the iron coming from the blast furnaces has the following average analysis: Total carbon, 4.25; silicon, 1.00; sulphur, 0.05; phosphorus, 1.50; manganese, 0.20 per cent. The iron is delivered from the blast furnace to the mixer, and from the mixer to the converters, by means of hot-metal ladle cars, and charged into the mixer with overhead traveling crane, and into the converters by means of an electric ladle hoist.

From 2,600 to 2,800 lb. of burned lime is charged into the empty converter, after which is charged about 11 tons of fluid pig iron from the hot-metal mixer. After the metal has been blown, the slag is skimmed into a cast-iron box car made for the purpose, and the metal is then poured into the ladle car, which transports it to the open-hearth furnace, the entire blow consuming from 12 to 15 min. Under good average conditions the blown metal has the following analysis: Carbon, 0.03; phosphorus, 0.07; sulphur, 0.05 per cent.; manganese, none; and the slag is constituted as follows: Silica, 13.0 to 14; alumina, 1.0; lime, 48.0 to 51; magnesia, 2.0 to 4; phosphoric acid, 17.0 to 19; manganous oxide, 1.5; iron protoxide, 13.0 to 15. Five ladles of the blown metal are charged into the open-hearth furnace as they are delivered from the converters, but, prior to this, the open-hearth furnace has been charged with about 4,000 lb. of burned lime and 6 to 8 tons of molten iron, direct from the hot-metal mixer, the latter iron being depended upon to give sufficient carbon for the chemical reaction. Ten to twelve heats are made in No. 1 open-hearth furnace in 24 hr. One of the economies resulting from the practice of the basic duplex process is the revenue derived from the Bessemer slag after it has been ground and prepared for agricultural purposes.

The Bethlehem Steel Co.'s Plant

In 1910 the Bethlehem Steel Co., desiring to increase and add flexibility to the output of its new Saucon plant and to render itself more independent with respect to the fluctuations of the scrap market, decided to adopt the duplex process, and, consequently, in 1911, a plant was installed. Rather than further burden its own organization, it let the contract to the Pennsylvania Engineering Works for the complete plant, which was designed, built, and turned over to it in operating condition. The

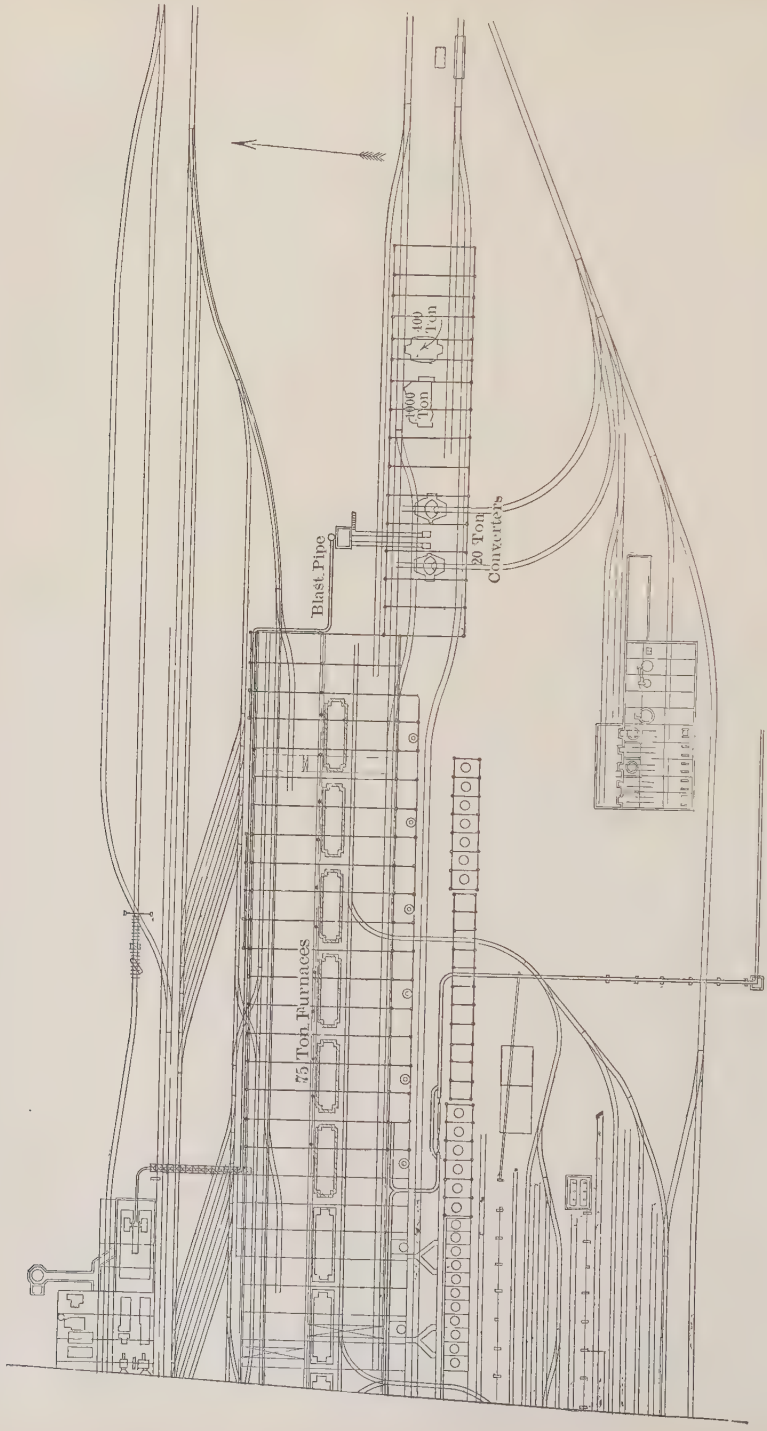


FIG. 3.—PLAN OF THE DUPLEX PLANT OF THE BETHLEHEM STEEL CO.

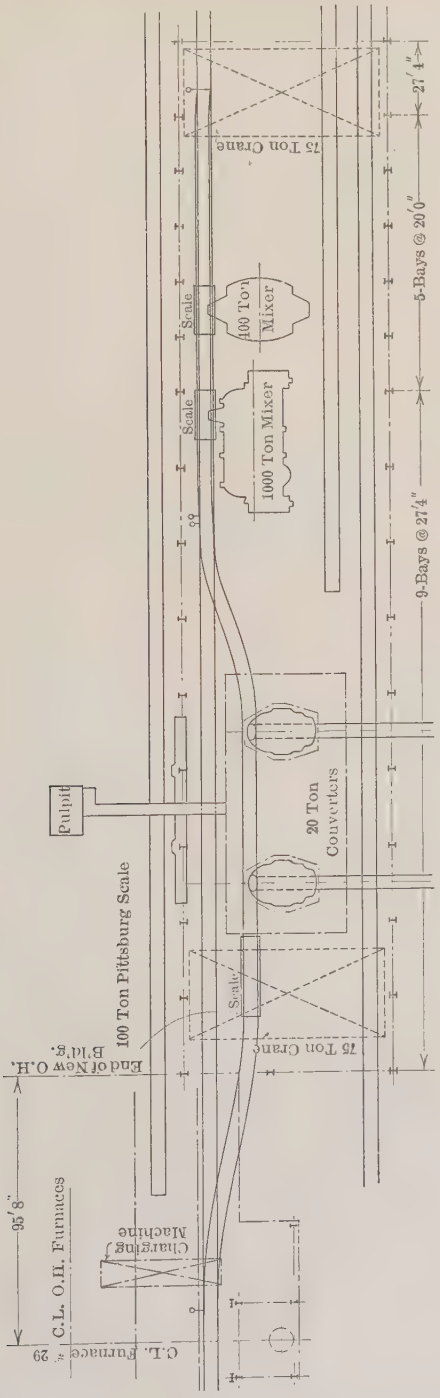


FIG. 4.—PLAN OF CONVERTER AND MIXER DEPARTMENTS, BETHLEHEM STEEL CO.

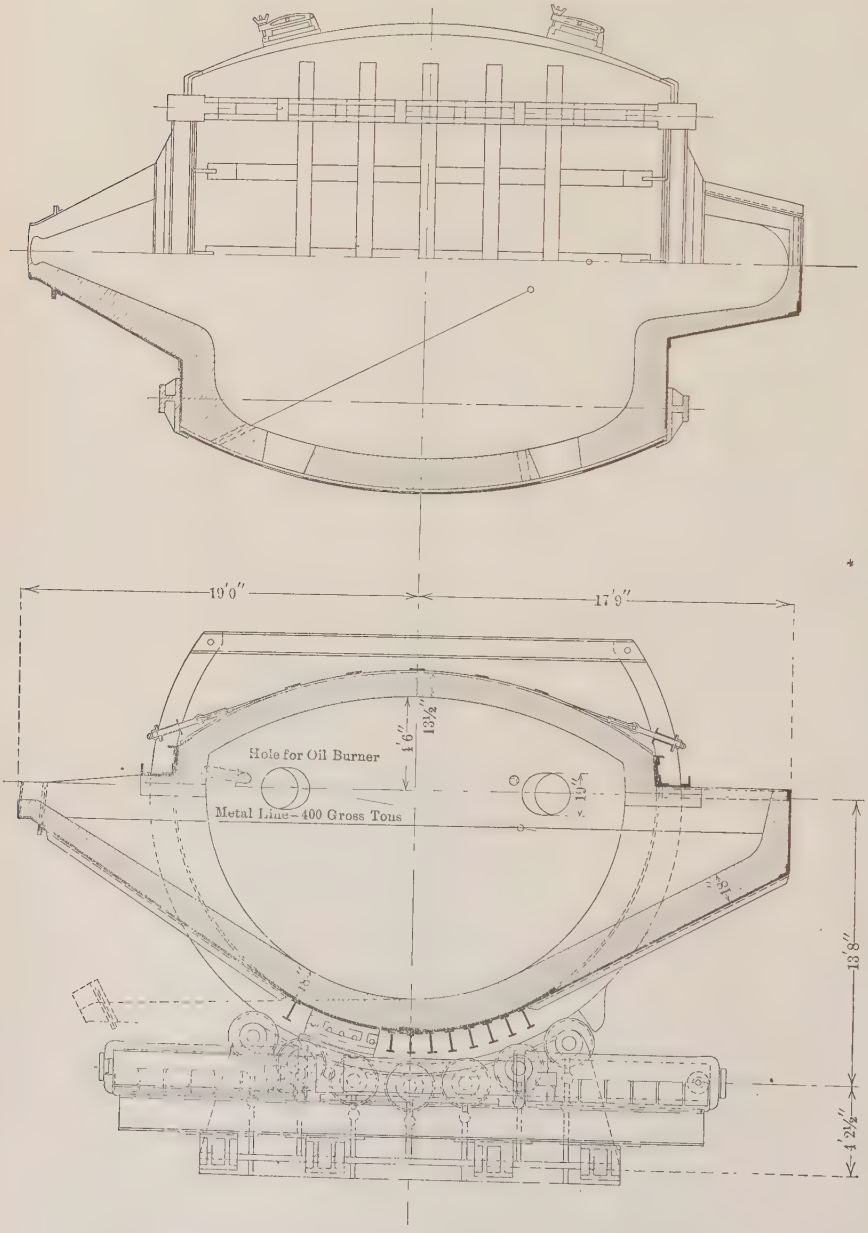


FIG. 5A.—400-TON HOT-METAL MIXER, BETHLEHEM STEEL CO.

original Saucon Steel Works, which were put into operation in 1907, consisted of ten 60-ton stationary open-hearth furnaces and one 250-ton hot-metal mixer, a Gray universal structural mill for rolling wide-flange I-beams and H-column sections, a standard structural shape mill, and a rail mill. In 1913, the Bethlehem Steel Co. added to the open-hearth department of the Saucon Works, six 75-ton open-hearth furnaces of the stationary type, locating them on the east end of the original open-hearth plant, and added to the mixer capacity one 1,000-ton hot-metal mixer of what is known as the German type. The Bessemer converting plant occupies a position east of, adjacent to, and in line with, the open-hearth building, thus making it possible to

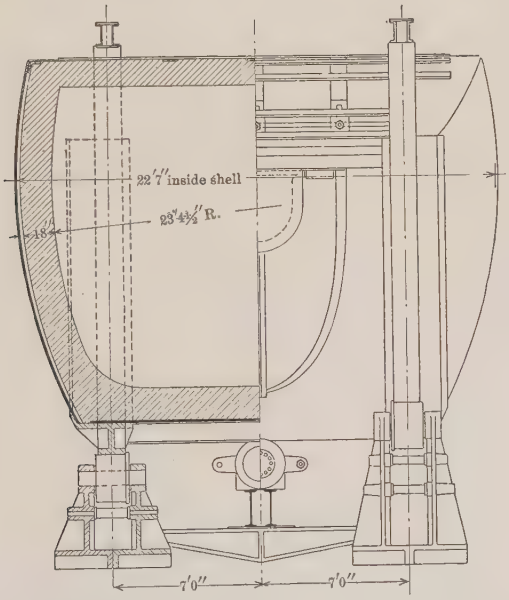


FIG. 5B.—400-TON HOT-METAL MIXER, BETHLEHEM STEEL CO.

connect the track for the transport of the blown metal from the converters to the open-hearth department directly with the track in the open-hearth building running parallel with the furnaces, but back of the charging-machine track. We might point out here that the location of the converters with respect to the open-hearth furnaces in this process is of the utmost importance, because the quick and easy transfer of the blown metal is one of the essential factors of success. The output of this department at present, operating on a straight open-hearth basis, is 70,000 to 75,000 tons of ingots monthly. As duplexing is practiced only part of the time, it cannot be said just what the output would be if the works were operated continuously under the duplexing method; but actual results have shown that for a given tonnage of steel there is a saving of about 65 per cent. in time.

The Bessemer converting plant, the general arrangement of which is shown in Figs. 3 and 4, is contained in two buildings, one known as the mixer-converter building, where the hot-metal mixers and converters are housed, and the other as the bottom house, where the bottoms are made up, dried, ladles repaired, etc. The floors of these buildings are on the same level as the charging floor of the open-hearth building. The mixer-converter building is of steel, and of the usual heavy mill type construction. It is provided with two 75-ton capacity, 70-ft. span, traveling cranes for serving the hot-metal mixers and converters. The converters occupy the west end of the building nearest the open-hearth plant, while the hot-metal mixers occupy the east end.

There are two hot-metal mixers, one of 400 tons capacity, of the type indicated in Fig. 5, and one of 1,000 tons capacity, of what is known as the German type. While these mixers have a combined nominal capacity of 1,400 gross tons, their actual capacity is considerably in excess thereof, since the larger mixer will hold at least 1,200 gross tons of molten iron. The 400-ton mixer was originally built to operate hydraulically, but has since been converted to operate electrically. It is mounted on heavy, cast-iron rocker stands, which furnish tracks for the cast-iron rollers. The two cast-steel rockers serve the dual purpose of a cradle for supporting the plate work and a tire for revolving the mixer on the rollers. The shell, which is semi-cylindrical in form with bulging convex ends and domed roof, with receiving and pouring spouts on the straight portion of the cylinder, is made of 1-in. steel plates. The mixer is lined throughout, to well above the slag line, with 9-in. thick magnesite brick, backed up by 9 in. of good grade fire brick, while the roof has a $13\frac{1}{2}$-in. lining of furnace-roof brick.

The 1,000-ton hot-metal mixer, constituting the other unit of this plant, is interesting on account of its large capacity as compared to former mixers in this country, 600 tons being the limit of capacity up to that time. At the time of making the contract for the building of the mixer. the Bethlehem Steel Co. stipulated that the builder should, before designing the mixer, send his engineers to Germany to investigate both the design and method of operation of the large mixers of that country, the result of which investigation was that the engineers came home and designed the present mixer after the German type, making such alterations and improvements as were found necessary to meet American conditions. (See Fig. 6.) The mixer consists of a cylindrical shell with spherical ends, It is provided with receiving spout on one side and pouring spout on the opposite side. The receiving spout, except when metal is being poured into the mixer, is sealed by a door, which is opened and closed by means by a $7\frac{1}{2}$-h.p. electric motor. The pouring spout is closed by means of a number of flat brick arches held in steel stirrups, except a very small opening for the egress of the metal, thus making it possible to conserve

from 90 to 96 per cent. of the original heat of the iron charged into the mixer.

A comparison might be made here between the two types of mixers: In the case of the 400-ton mixer it will be noted that the area of the surface of the metal is large compared with the depth and total volume of the metal, whereas, in the 1,000-ton mixer, the area of the surface of the metal is small as compared to the depth and total volume. This feature also

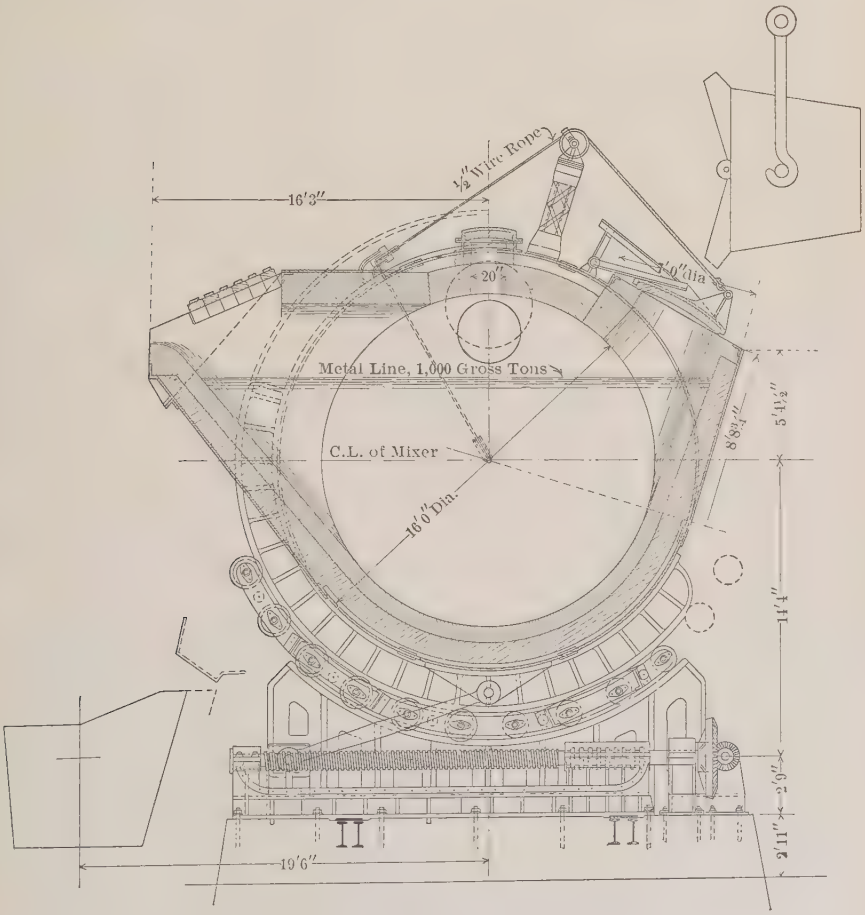


FIG. 6.—1,000-TON HOT-METAL MIXER, BETHLEHEM STEEL CO.

has its bearing on the conservation of heat. The 1,000-ton mixer is provided with a heating apparatus in which oil, producer, coke-oven, or blast-furnace gas is used. During the investigation of mixers in Germany it was found that, while most of the mixers were provided with a means of artificial heating, nevertheless the artificial heat was unnecessary except in one instance, and, in most cases, the ports for this heat entry were

blanked off. At South Bethlehem it has been found to be of advantage, however, to use a small amount of artificial heat. The cylindrical shell of the 1,000-ton mixer is provided with four cast-steel bands, spaced uniformly between the two ends. The two outer bands encircle the cylinder about half way, while the two inner bands form a complete circle. These bands serve the dual purpose of reinforcing the shell and supplying tires on which the mixer rotates on the rollers. There are four groups of 11 cast-steel rollers, 16 in. in diameter with 16-in. tread. Each group of rollers is formed into a cradle by means of heavy side bars, with shafts extending through the side bars and rollers, thus insuring equal spacing of the rollers in the groups at all times. Two lines of heavy struts are provided between the cradles, to insure the action of all the groups as one unit. The weight of the mixer is transmitted directly from the rollers to four heavy cast-iron roller stands, thus producing true roller bearings, and minimizing the friction. It may be worthy of mention here that, with this type of mixer, the center of rotation coincides with the center of gravity, whereas, in the former type, it is necessary to lift the center of gravity about the center of rotation, from which it will easily be seen that the larger type of mixer requires, relatively, considerably less power to operate. In order to insure alignment of the rocker stands, and to tie the mixer more firmly to the foundations, two heavy I-beam girders are provided, extending longitudinally under the rocker stands the full length of the mixer. These girders are provided with shear plates, while the bases of the stands are provided with shear lugs. This feature is also of considerable advantage when lining up the mixer at the time of erection. The 1,000-ton mixer is operated by two 75-h.p. electric motors, either of which is capable of furnishing the required power. The motors are controlled by a magnetic switch-type controller, and are arranged so that they will operate in series during the pouring period and in parallel during the return of the mixer to its normal position, thus reducing the time for pouring and the return of the mixer to a minimum. The motors are placed one at either end of the mixer and are both connected to the line shaft, which is provided with a clutch adjacent to either motor, which may be disengaged in case of accident or for repairs. The 1,000-ton mixer has a surface lining 9 in. thick of magnesite fire brick extending well above the slag line, which is backed with $13\frac{1}{2}$ in. of fire brick, which, with the packing between the lining and the shell, makes a total thickness of lining of about 2 ft. The extraordinary thickness of the lining also tends to conserve the heat of the mixer contents.

The converters, which are of 20 tons capacity, are spaced 40 ft. center to center, and are similar to those described at the plant of the Tennessee Coal, Iron & Railroad Co., with the exception that in this case it was thought desirable to contract the body of the vessel where it joins the bottom, in order to make the diameter of the vessel less, thus bringing

the tuyères nearer to the side walls. This has a tendency to deeper the bath slightly for a given charge of metal; but, on the other hand, brings the bath more directly over the tuyère openings. The converters are operated by means of hydraulic cylinders through racks and pinions, similar to those of the Tennessee Coal, Iron & Railroad Co. The hydraulic cylinders are on a pressure line of 550 lb., which is provided with tank accumulator to balance the pressure. The converters are operated by means of Aiken valves, and the pressure lines throughout are made of double extra pipe. The blast for blowing the converters is furnished by Southwark horizontal cross-compound engine with barometric condenser, with steam cylinders 46 and 84 by 60 in. and air cylinders 84 by 60 in. The engine is designed to furnish 45,000 ft. of free air per minute, at 30 lb. pressure, and is located in the power house adjacent to the open-hearth

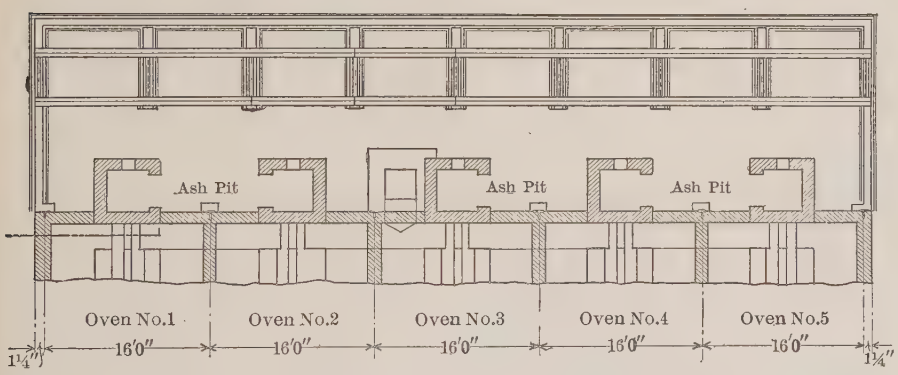


FIG. 7.—PLAN OF BOTTOM OVENS, BETHLEHEM STEEL CO.

plant. The blast is conducted to the converters through a 30-in. main about 500 ft. in length, which is provided with air-relief valve to prevent back pressure to engine. It is also provided with a tank air receiver near the converters to balance the pressure. The air is delivered at a pressure of 18 to 20 lb. from the receiver tank to each converter through an 18-in. air line provided with an 18-in. air valve operated from the pulpit. Heavy steel construction operating floors and platforms are provided about both the mixers and the converters. One 75-ton track scale is provided directly under the pouring spout of each mixer, and another between the converters and the open-hearth furnaces, on which the metal is weighed prior to charging into the converters and again after it has been blown, on its way from the converters to the open-hearth department.

The bottom house is located just south of the mixer-converter building. It has a 16-ft. leanto extending throughout the full length of the building, which is divded into ten equal bays. Five of these bays in the leanto are occupied by the five ovens for drying the bottoms. The ovens, shown in Figs. 7 and 8, have flue connection with a common draft stack, and are

heated by firing from the rear with small anthacite coal, for which special grates are provided. Forced draft is furnished by an Eynon & Evans blower on each fire box. The ovens are equipped with Kinnear roller curtain doors and each one has a floor space of 14 by 16 ft. This house

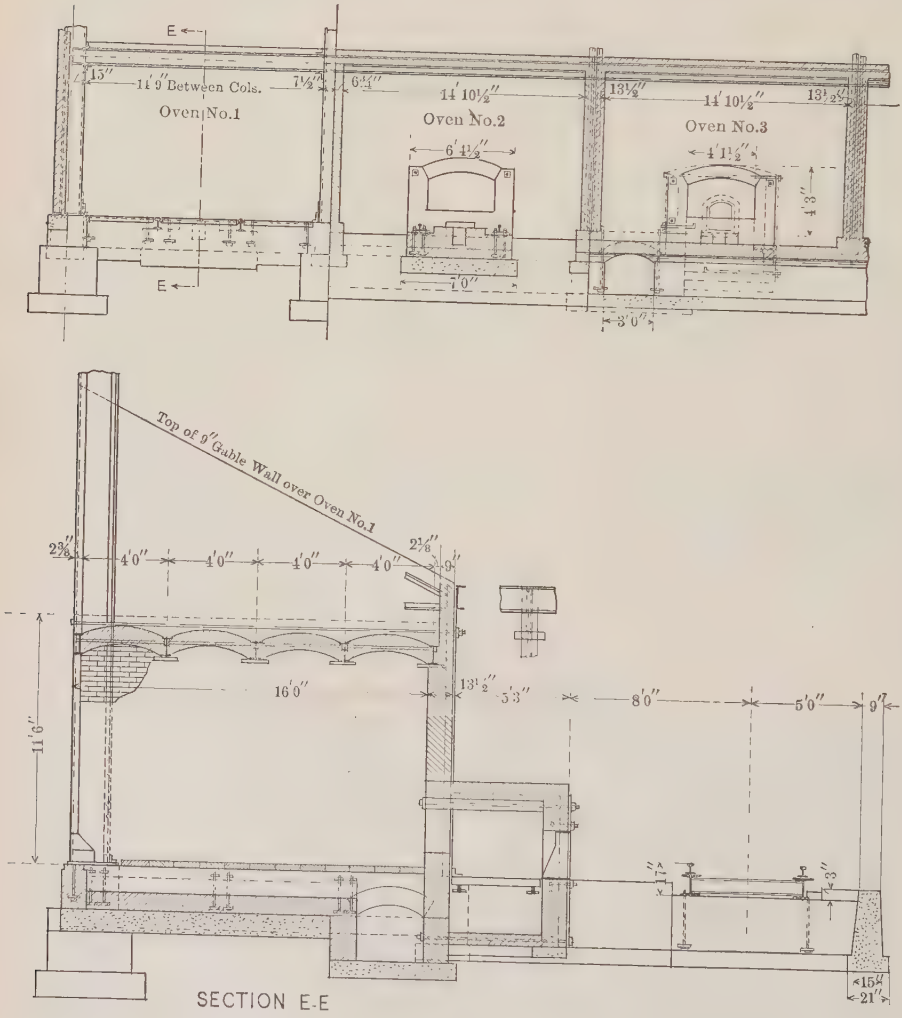


Fig. 8.—Elevation and Section of Bottom Ovens, Bethlehem Steel Co.

contains a bottom pit, over which the bottoms are made up, a Blake crusher for crushing stone for linings, an 8-ft. wet pan, a 9-ft. dry pan, and the necessary bins and conveyor for handling the raw material. The machinery is driven by a 100-h.p. electric motor. There are also ten ladle rests for the repair of the linings of the steel ladles. For lifting

the bottoms on and off the cars and bottom pit, handling the ladles, etc., there is provided a 25-ton electric traveling crane. The burned-out bottoms are brought from the converters, and the newly made up bottoms returned to the converters, on hydraulic jack cars.

The method of operation at the Saucon Works is practically as follows: The molten pig iron is brought from the blast furnaces situated at the Lehigh Works, 1½ miles distant, in trains of 40-ton capacity hot-metal cars drawn by the usual standard-gauge locomotive. Just before entering the mixer building, the metal is weighed on a 100-ton capacity track scale. It is then drawn into the building directly opposite the mixers and is poured into the mixers by means of the 75-ton overhead electric traveling crane. The metal is poured from the mixer into a 25-ton capacity ladle car, weighed on the 75-ton track scale placed in the floor directly under the pouring spout of each mixer, and then transported to the converter, where it is charged by means of the overhead traveling crane. The ladle is of special design with respect to the pouring spout, the object being to retain as much of the slag as possible in the ladle. For this purpose the spout is made so that the metal will pour through a narrow and rather deep opening. The slag is retained in the ladle for several charges and then dumped out. At some plants skimming is resorted to. This is especially true in Germany, where the ladles of metal are skimmed both before pouring into the mixer and before charging into the converter. After the metal has been blown in the converter, it is poured into 25-ton ladle cars (two of which are in constant use for the transfer of metal when duplexing), and transported by an electric locomotive over a standard-gauge track to the open-hearth furnaces. The hot metal is poured by the overhead traveling crane into the furnace, through a portable spout, which is placed in position by the charging machine at time of charging. Five of the ten 60-ton stationary open-hearth furnaces are used in the duplexing process; the remaining five 60-ton furnaces and the six new 75-ton furnaces are kept on the straight open-hearth practice making steel from the hot-metal mixer iron and scrap. Ordinarily three ladles of converter iron are charged into the open-hearth furnace, one after another, as rapidly as they can be blown. All of the metal is desiliconized in the converter and, of the three ladles of metal constituting the open-hearth furnace charge, the former two have practically all of the carbon eliminated, while, in the last one, about 2 per cent. of carbon is left, to bring about the reaction in the open-hearth furnace. The average open-hearth furnace charge is 40,000 lb. scrap, 15,000 lb. burnt lime, and 95,000 lb. of converter iron. The time required in the open-hearth furnace varies in general practice from 4 to 6 hr., although a single heat has been put through in 3½ hr. At times, recarbonizing is found necessary, and, for this purpose, there is in the open-hearth building a 250-ton hot-metal mixer employed as a receiver for the recarbonizing iron, which is of a special

Bessemer quality made from low-phosphorus ores. This metal is poured from the mixer into ladles, and added to the bath in the open-hearth furnaces, as needed.

The Pennsylvania Steel Co.'s Plant

The steel-making department of this company, in which duplexing is carried on, consists of six 75-ton and two 200-ton open-hearth furnaces, two 20-ton Bessemer converters, one 300-ton and one 800-ton hot-metal mixer, and a bottom house equipped with the necessary crushing and grinding machinery, drying ovens, etc., for preparing the material, making up, and drying the bottoms. The plant was built in 1913, with the exception of the six 75-ton open-hearth furnaces and the 300-ton hot-metal mixer, which, previous to this time, were run on straight open-hearth practice. The contract for the 800-ton mixer and the two converters was let on April 19 and the converters blown in on the following December 1, making a record time of $7\frac{1}{3}$ months for the construction of such a plant. As indicated in Fig. 9, the mixer-converter department is situated in the main open-hearth building on the charging-floor side. Extending round about both the mixers and the converters there is a spacious working platform of steel construction, 20 ft. above the ground floor and at the same elevation as the open-hearth charging floor. The molten pig iron, coming from the blast furnaces in trains of 45-ton hot-metal cars, is poured direct into the receiving spouts of the mixers without removing the ladles from the cars. The mixers pour into ladles resting on the ground floor, which are hoisted and charged into the converters by the overhead traveling crane. The converters in turn pour into ladles resting on the ground floor, which are hoisted to the open-hearth charging floor and transported by the overhead traveling crane to the open-hearth furnace to be charged.

The hot-metal mixers are of the semi-cylindrical, bulging-end type, and are tilted by electric motors operating through suitable trains of gears and screws. The control equipment is of the magnetic switch type. The converters, shown in Fig. 10, are placed 42 ft. center to center. They are duplicates of those of the Bethlehem Steel Co., with the exception of the overturning arrangement, which is electrical. Each converter is provided with two 100-h.p. motors controlled by magnetic switch-type controller. Either motor is capable of operating the vessel and there are shaft couplings provided so that either motor may be disengaged at any time. The power is transmitted from the motors through one gear reduction on the motors, a worm and worm wheel and pinion on the worm-wheel shaft to the driving gear on the converter trunnion. Mention is made here of the worm wheel, which has the wearing faces of the teeth lined with $\frac{5}{16}$ in. of babbitt metal, which reduces the friction, thus prolonging the life of the teeth. The arrangement of this plant deserves special attention, as

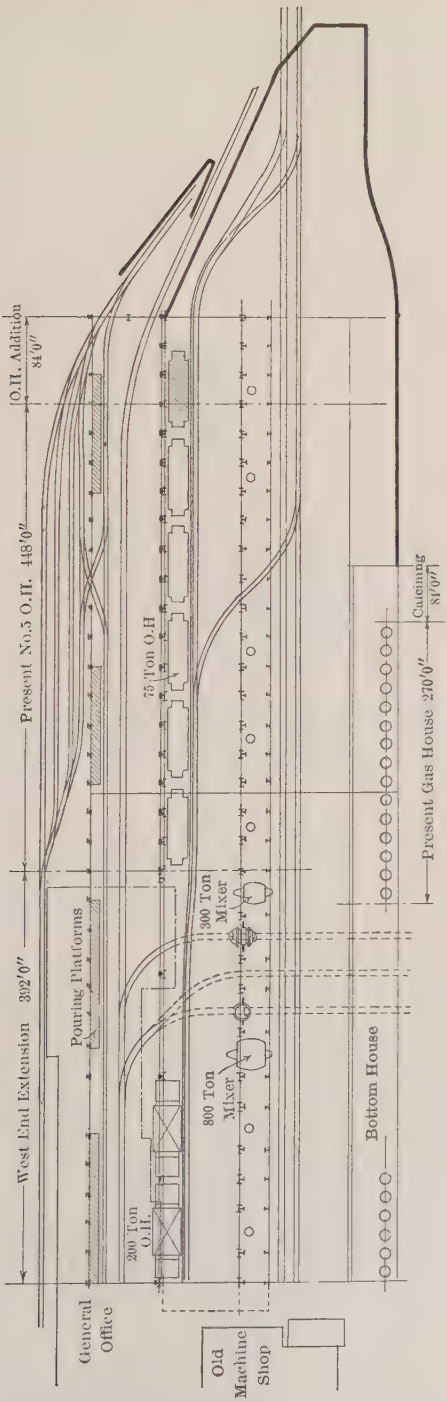


FIG. 9.—PLAN OF THE DUPLEX PLANT OF THE PENNSYLVANIA STEEL CO.

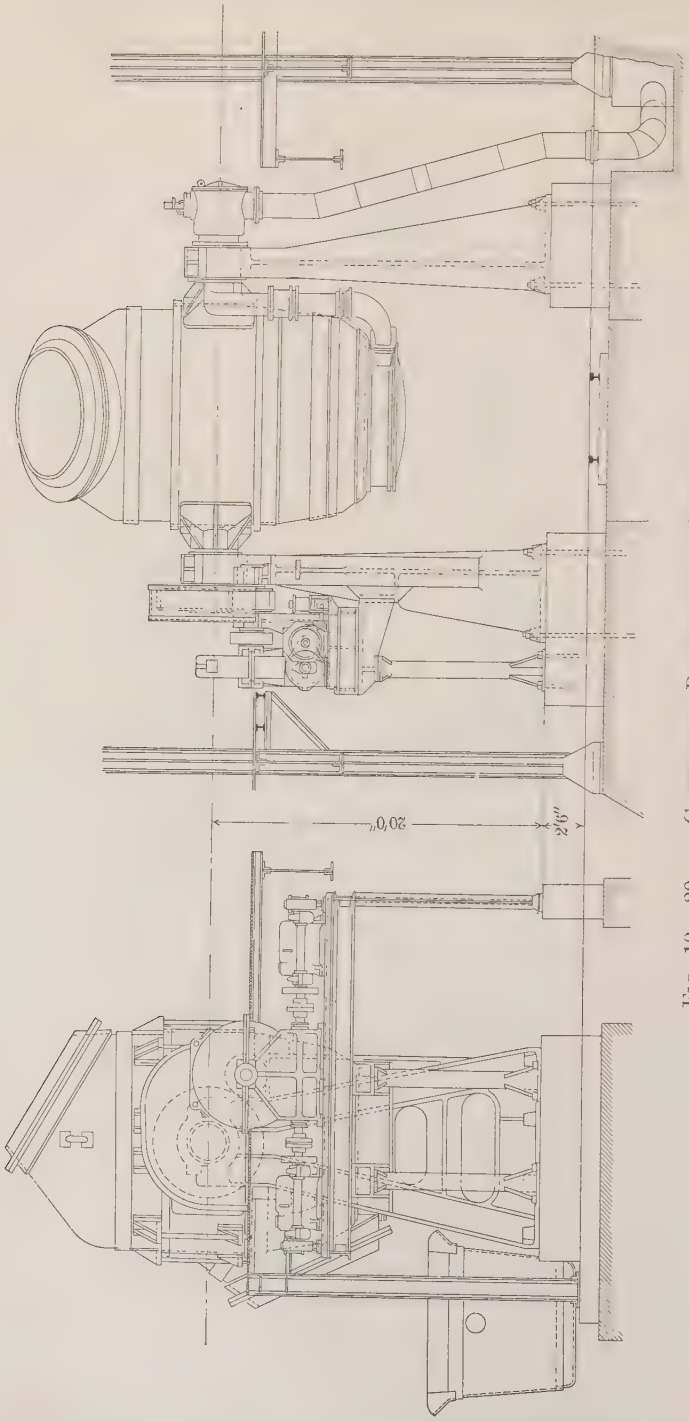


FIG. 10.—20-TON CONVERTER, PENNSYLVANIA STEEL CO.

it represents one of the best conditions for economy and convenience of operation which has been attained up to the present time.

General Remarks

The two 20-ton vessels as installed in the plants of the Tennessee Coal, Iron & Railroad Co., the Bethlehem Steel Co., the Jones & Laughlin Steel Co., and the Pennsylvania Steel Co., are capable of producing 100,000 tons of blown metal per month. The benefits derived from the duplex process for making steel have been enumerated in the reasons given for duplexing; but we may say in conclusion that the process has brought about the development of the Bessemer converting plant to a higher state of perfection and efficiency, has created a desire for larger and better hot-metal cars and mixers, and has aided those practicing it to solve some of their problems. Since the process has been practiced in this country for seven years only and is still in an imperfect state, it is hoped that some of the master minds working on it will succeed in bringing about a more perfect stage of development within the near succeeding years, and thus another step will have been taken toward the uplift of mankind.

Discussion

W. McA. Johnson, New York, N. Y.—On discussing this paper I am open to the criticism of "bringing coals to Newcastle," as I, a zinc man, am bringing new ideas on steel to Pittsburgh. But still the view of an outsider can be valuable to the insider. Any multistage process, where each apparatus operates on its particular part of the work at a high efficiency, constitutes an advance in the state of art. But in multiplicity there can be complication and ensuing loss and inefficiency.

The duplex steel process is not the pronounced success which was predicted for it by its adherents, for several of such plants have reverted to plain open-hearth methods.

If we regard broadly refining processes, such as the Bessemer, open-hearth, puddling, copper refining, lead cupelling, nickel refining, we find that in general refining processes are intermittent and concentration processes are continuous. (This generalization I believe to be original with me; it was communicated to Dr. E. F. Roeber, editor, and appeared as an editorial in *Metallurgical and Chemical Engineering*, March, 1905.)

Where we desire primarily quality in a metallurgical operation we must put a batch of metal in the furnace, work on it until we get it to the required purity, test it, and take it out; whereas, if we desire primarily quantity we shove ore into a furnace and smelt it as fast as possible, allowing conditions to make a non-uniform product provided only we get tonnage.

From general commercial principles, I believe that the weak point of

the duplex process is that the operation is not a money maker unless the plant is kept at 90 to 95 per cent. of its rated capacity, or unless the Bessemerizing is done so often that it is practically continuous. For in some manner the operation should be made continuous and the control on the tonnage should be commercially flexible. The commercial rigidity of having to control purity exactly increases operating costs without a corresponding gain. Conversely, the chief reason for failure of "steel from ore direct" is that it makes a final operation continuous, which should be naturally intermittent.

Arthur G. McKee, Cleveland, Ohio.—I would like to ask a couple of questions in regard to this paper, and in asking them I appreciate the difficulty that Mr. Furst has encountered in preparing a paper of this sort and in getting the information, which really, to my mind, is the important information to the owner of a plant for the producing of steel, and also to the man who is responsible for the building of such a plant.

In the first sentence is stated:

"The reasons for manufacturing steel by the duplex process are, briefly: saving of time; increasing output for capital invested; and avoiding the difficulty sometimes experienced in obtaining scrap."

That is briefly stated, surely, but how effective is such a plant in giving the results referred to in that statement?

At the close of the paper, Mr. Furst remarks:

"The benefits derived from the duplex process for making steel have been enumerated in the reasons given for duplexing; but we may say in conclusion that the process has brought about the development of the Bessemer converting plant to a higher state of perfection and efficiency, has created a desire for larger and better hot-metal cars and mixers, and has aided those practicing it to solve some of their problems. Since the process has been practiced in this country for seven years only and is still in an imperfect state, it is hoped that some of the master minds working on it will succeed in bringing about a more perfect stage of development within the near succeeding years, and thus another step will have been taken toward the uplift of mankind."

Does the process make, as claimed, a larger tonnage per day, and approximately how much? Does it reduce the cost per ton? Does it increase the production per dollar of plant investment, and if so, how much? And, also, another very pertinent question: How does this process affect the quality of the product, if at all?

Mr. Furst doubtless does not have an opportunity to get these facts, but can somebody else enlighten the Institute, so that we can know from actual experience in operation what conditions justify the installation of the equipment required for the use of this process; what profit and other advantages will be obtained; in short, whether it has justified the claims made for it.

Henry D. Hibbard, Plainfield, N. J.—Regarding Mr. Johnson's remarks, when you come to make a finished product, such as steel, you

want to take a batch of material and keep it in hand until it is in proper shape and then cast it. Where you are running for a crude product, as in the case of the blast furnace, it is all right to run continuously, but when you want a finished product, to try to avoid taking a batch and keeping it under control until it is done, will be a mistake. In the open-hearth furnace we take a charge and keep it until it is done; in the Bessemer, we have, to some extent, to catch it on the fly; but in each of these cases the charge is treated until it is as near as we can get it to what we want when it is cast. Ever since the Bessemer process was established, there have been proposals to make steel by treating iron with blast as it moved forward, and some of the pyrotechnic displays occasioned thereby have been very wonderful. Such plans are all failures. Within a few years patents have been taken out for processes of that description, but I think no progress has been made with any of them.

As to the output of the duplex process, I think it varies from about three open-hearth heats of finished steel to about 20 heats in 24 hr. There are furnaces in this country that have produced over 1,000 tons of steel a day and have kept it up for a month. On the other hand, three open-hearth heats of 50 tons each would give an output of 150 tons a day.

Bradley Stoughton, New York, N. Y.—As to the question whether the duplex process increases the output, it is possible to answer this in either of two ways:

The output of the duplex process, when properly operated, is much more than the output of the open-hearth process alone, but is not so great as the combined output of the open-hearth process and the Bessemer process each working independently of the other. This, however, is not the point really involved, because the output of the two processes working independently has now no longer the same industrial possibilities, because we have not in this country sufficient Bessemer ore to operate the Bessemer process economically and up to its capacity. But when the two processes work together they can use ores now available in this country in very great quantities, which are above the Bessemer limit, and can produce a good quality of steel.

As to the question whether the interest on the investment required to install the duplex process is more than compensated for by the increased profits, we can answer this by inference through our knowledge that several companies which are wisely managed, and which years ago abandoned the Bessemer process, have recently re-installed converters in order to employ them in the duplex process.

I believe that the duplex process has very decided commercial and industrial advantages, and hope that those who are operating it will cooperate with this Institute by coming forward and exchanging their information with others in order that benefit may be accorded to all.

Sound Ingots

BY SIR ROBERT HADFIELD,* F. R. S., LONDON, ENGLAND

(New York Meeting, February, 1914)

LAST year this Institute was good enough to accept some remarks by the writer regarding sound steel, entitled Plant for Hadfield Method of Producing Sound Steel Ingots, being a continuation of a research in which for some time the writer has been engaged, and full particulars of which were submitted in papers entitled A New Method of Revealing Segregation in Steel Ingots, and Method of Producing Sound Ingots, read before the Iron and Steel Institute in October, 1912. These papers gave rise to a very interesting discussion, both in England and America.

This question is one of vital importance, as unless sound steel in the form of ingots is first obtained, sound steel in the finished forms of rail billets, rails, bars, sheets, and other forms cannot be expected. The working stresses demanded by modern conditions are so great, each year they probably approach nearer to the limit of safety, that without doubt those who produce steel will be compelled to pay the attention which this subject deserves. Moreover, the problem is not an insuperable one, as shown by the writer in the papers above referred to.

In view of the information which has already been presented in these papers, it is not now necessary to go into details of the system advocated by the writer, except to repeat that the ingots and steel now referred to were produced by employing in the first place sound fluid steel, free from blowholes, this being poured into ordinary ingot molds, provided with the necessary feeding heads, the insulating slag medium, also charcoal and air blast or other suitable form of applying heat to the head portion of the ingot. The upper or head portion of the ingot is kept fluid by the intense heat generated by the air blast and charcoal, so that such fluid steel passes into the body of the ingot below the head, thus feeding and avoiding the shrinkage or piping which would otherwise occur. The result is the production of a sound steel ingot, free from blowholes, segregation and piping, and in which the waste portion, usually termed the discard, is but little over 7 or 8 per cent.

Billets produced from such ingots have shown a salable product of about 91 per cent., including the oxidation losses in heating and rolling. Plate I shows the details of the products obtained from such an ingot. While these results were obtained from a comparatively small ingot, 11 in.

* Honorary Member.

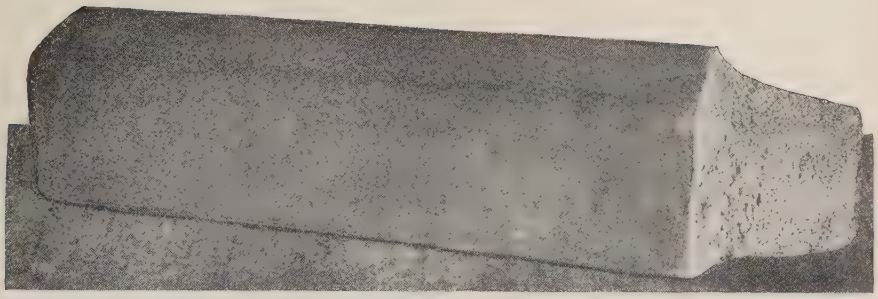


FIG. 1.—11-in. Ingot, 4 ft. 4 in. long, weighing 1,680 lb.

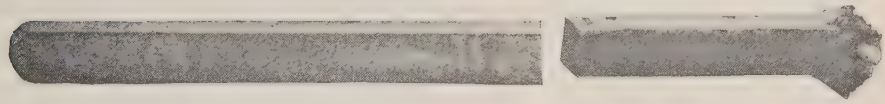


FIG. 2.—Billet from top portion of Ingot.

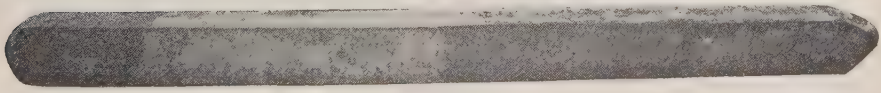


FIG. 3.—Billet from bottom portion of Ingot.

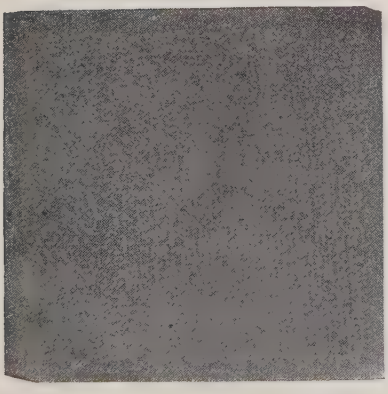


FIG. 4.—Section at *A* of Fig. 2. Sound metal.

FIG. 5.—Section at *B* of Fig. 2, 1 in. from *A*. Last trace of piping.

PLATE I.

square, weighing 1,680 lb., the same results have been obtained from 14, 15, 18, and 20 in. ingots.

G. Charpy of the Cie. des Forges et Acieries de Chatillon-Commentry et Neuves-Maisons, Montluçon (Allier), also made an ingot weighing 25 tons under this system, and reported to the writer the following results with regard to the system as applied to larger ingots.

M. Charpy stated that the Hadfield method of feeding the upper portion of the ingot in the special manner described in the papers mentioned above, namely, sand head, air blast, charcoal, and insulating layer of slag, appeared to him to constitute a particularly simple and practical solution of the problem met with in steel manufacture, and to overcome the difficulties met with as regards piping; also to reduce segregation and avoid the large percentage of waste material.

In Plate II, Fig. 1 represents a photograph taken by M. Charpy of the upper part of an ingot weighing 25 tons. This ingot has been treated by the writer's method, its composition being: Carbon, 0.40; sulphur, 0.02; phosphorus, 0.04; manganese, 0.45 per cent. The material also contained a small percentage of nickel.

Fig. 1 shows the satisfactory nature of the method. The portion of the ingot shown in the photograph represents about one-fourth of its total length. M. Charpy found that in order to eliminate all faults and sponginess in the ingot in question, it was only necessary to cut off about 12 to 12½ in. from the upper part; in other words, about one-twentieth of the ingot, as against one-third which is the proportion of discard ordinarily required. The waste and segregated material in this ingot only amounted to about 5 per cent. of the total weight.

Moreover, the analyses taken of the material at different parts of the ingot show this is free from segregation.

Fig. 2 shows the parts from which the analyses given in the table were taken.

Drillings were taken on the center line at the different parts shown, commencing at the top, marked *A*, also at the other parts as lettered.

Portion Drilled	S Per Cent.	P Per Cent.	Portion Drilled	S Per Cent.	P Per Cent.
A	0.016	0.043	K	0.012	0.044
B	0.018	0.041	M	0.018	0.039
C	0.028	0.043	N	0.017	0.039
D	0.024	0.042	O	0.026	0.044
E	0.021	0.039	P	0.028	0.045
H	0.020	0.043	R	0.015	0.038
I	0.020	0.039	X	0.015	0.038

FIG. 1.—Section of upper part of 25-ton Ingot.

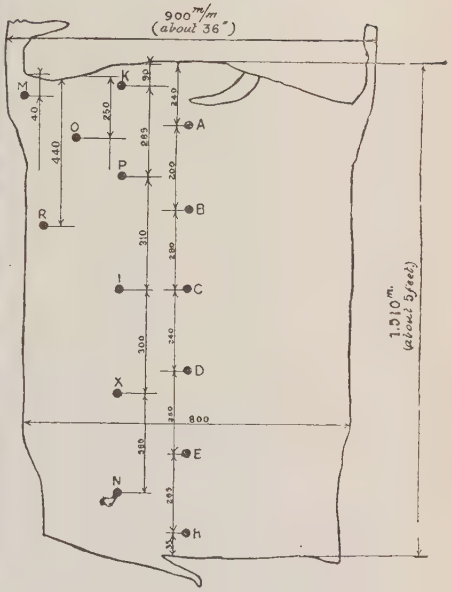


FIG 2.—Showing parts drilled.

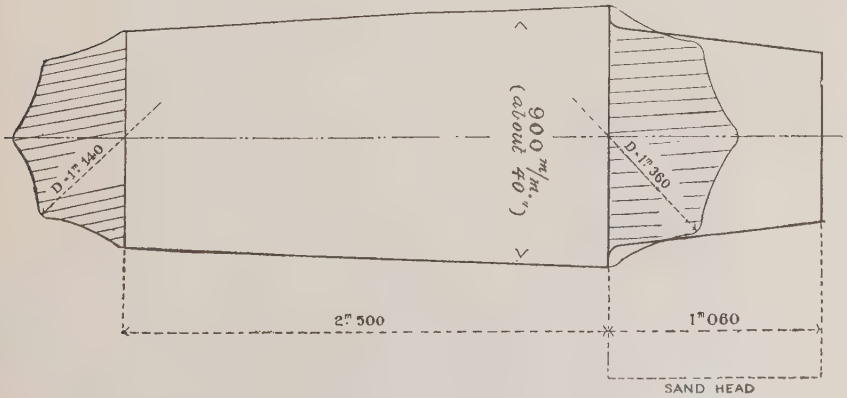


FIG. 3.—Dimensions of 25-ton Ingot.

PLATE II.

Fig. 3 shows the dimensions of the ingot.

With reference to the question of soundness of material, it may be interesting to point out that in the smaller ingot shown in Plate I, Fig. 1, from which the billets were obtained, Figs. 2 and 3, even including the sand head portion (termed head or discard in Plate I), the total head or discard was only 2 ft. 5 in. in length. By cutting off 1 in. more, Fig. 4 shows the polished section from the top portion of this billet at the part marked *A* in Fig. 2; then comes the length of 1 in. removed; above this is the head or discard mentioned, 2 ft. 5 in. in length.

Fig. 5 represents the polished section from the top portion of the billet at the part marked *B* in Fig. 2. In other words, Fig. 5 shows the last traces of piping near the head, at the part marked *B* in Fig. 2, while Fig. 4, 1 in. further down, shows the complete removal of this piping, represented by the polished section from the top portion of the billet at the part marked *A* in Fig. 2. That is to say, the material represented in Fig. 4 and that below this represents sound material.

Fig. 5 also represents the head or discard of this ingot. The ingot as cast weighed 1,680 lb.; the discard removed weighed 123 lb., or 7.3 per cent. of the total weight of the ingot. The total weight of sound billet was 1,513 lb. or 90 per cent.

In a similar manner, from another ingot there was obtained a sound rail having a total length, wholly free from pipe, of 62 ft. 7 in., with a discard of only 18 in. of rail, and in addition to this, the ingot head itself, representing 7 to 8 per cent. of the weight of the ingot.

In order to further illustrate this method, Plate III represents four ingots, *A* and *C* made by the writer's system, *B* and *D* ingots made in the ordinary way. Ingot *B* was cast from slightly rising steel, and ingot *D* from piping steel but not fed. Ingots *A* and *B*, also *C* and *D*, were cast from the same heats respectively. These ingots are also shown in vertical position in Plate IV.

Another illustration is shown in Plates V and VI of ingots made in the ordinary way. These represent 18-in. ingots, weighing about $2\frac{3}{4}$ tons each, cast with the small end up, *A* being rising or unsound material, and *B* piped material but not fed.

In comparison are shown Plates VII and VIII, which also represent the same sized ingots, namely 18 in., made by the writer's system, weighing about $2\frac{3}{4}$ tons each. These are interesting because the ingots were cast with the small end up, as in ordinary practice. Plate VII gives an excellent view of the cavity produced by the sound steel in these ingots as it settles down into the body of the ingot proper, that is below the feeding head. As will be seen, by a merely superficial examination of such an ingot, occupying but a few seconds, it can be determined whether the steel in the ingot itself is sound or not. If the cavity is there, the material below in the ingot proper will be perfectly sound and free from honeycombs, blowholes, segregation, or piping.

A and *C*, Hadfield system. *B* and *D*, ordinary method.

PLATE III.—15-in. Ingots, weighing about 2 tons each.

It will be admitted by any impartial observer that it is, on the other hand, quite impossible, by looking at either of the ingots shown in Plate V, made in the ordinary manner, that is without the head or feeding arrangement, to determine whether these are sound or unsound, or the nature of the unsoundness or piping present in them, that is below the top of the ingot. In other words, the ingots would have to be sliced

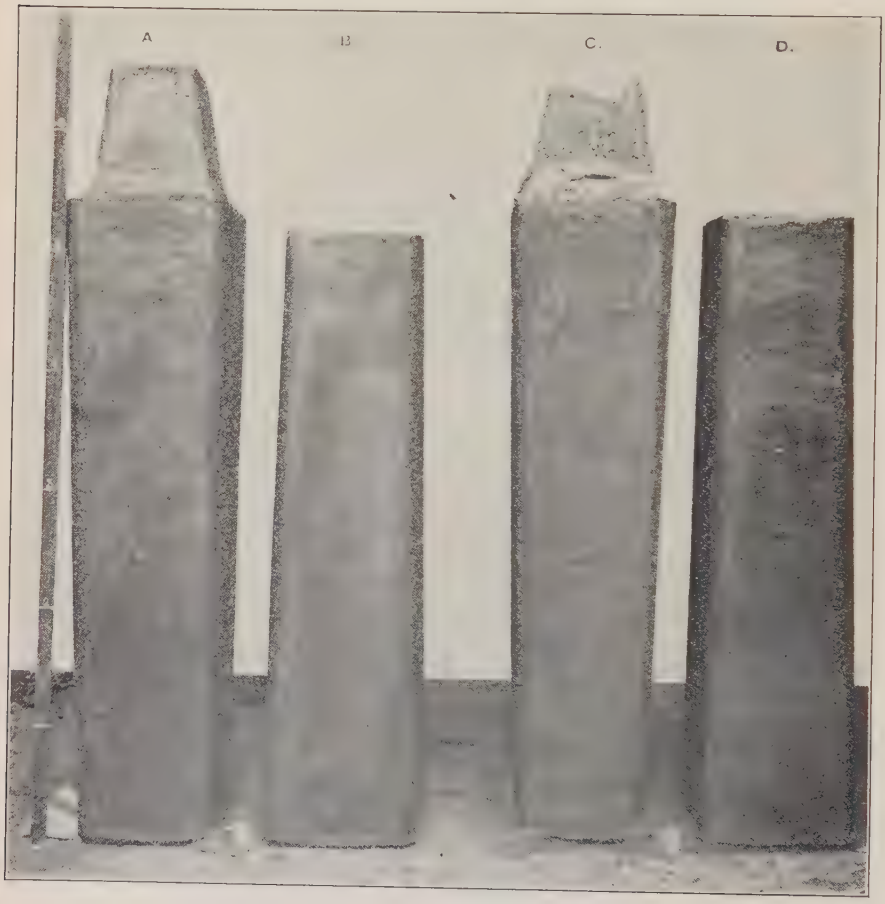


PLATE IV.—ANOTHER VIEW OF THE INGOTS SHOWN IN PLATE III.

up by machining, and thus spoiled for actual work, before it could be decided whether they would give perfectly sound material, whereas in the ingots shown in Plate VII, made under the writer's system, machining or other observation carried out by mechanical methods is entirely unnecessary, and at least 88 to 90 per cent. of sound usable and salable material is obtained. This, too, can be accomplished just as readily and easily with 10,000 as with one ingot.

A, Unsound Ingot. *B*, Piped Ingot.

PLATE V.—ORDINARY SYSTEM 18-IN. INGOTS, WEIGHING ABOUT 2¾ TONS EACH.

Moreover, while the two ingots shown in Plate IV possess some difference, the "rising" ingot, shown by *A*, is probably the better, because by a superficial examination it can be at once determined that such material is unsound. While rolling or forging such an ingot may help partly to

PLATE VI.—ANOTHER VIEW OF THE INGOTS SHOWN IN PLATE V.

close up not all but some of the blowholes, yet it is not easy to believe such material is as perfect or possesses the same mechanical strength as that resulting from an ingot which has been cast from steel containing no blowholes. Imagine such an ingot as represented by *A* in Plate V being used to produce, say, a gun forging! This would not be allowed by

PLATE VII.—HADFIELD SYSTEM 18-IN. INGOTS, WEIGHING ABOUT 2¾ TONS EACH.

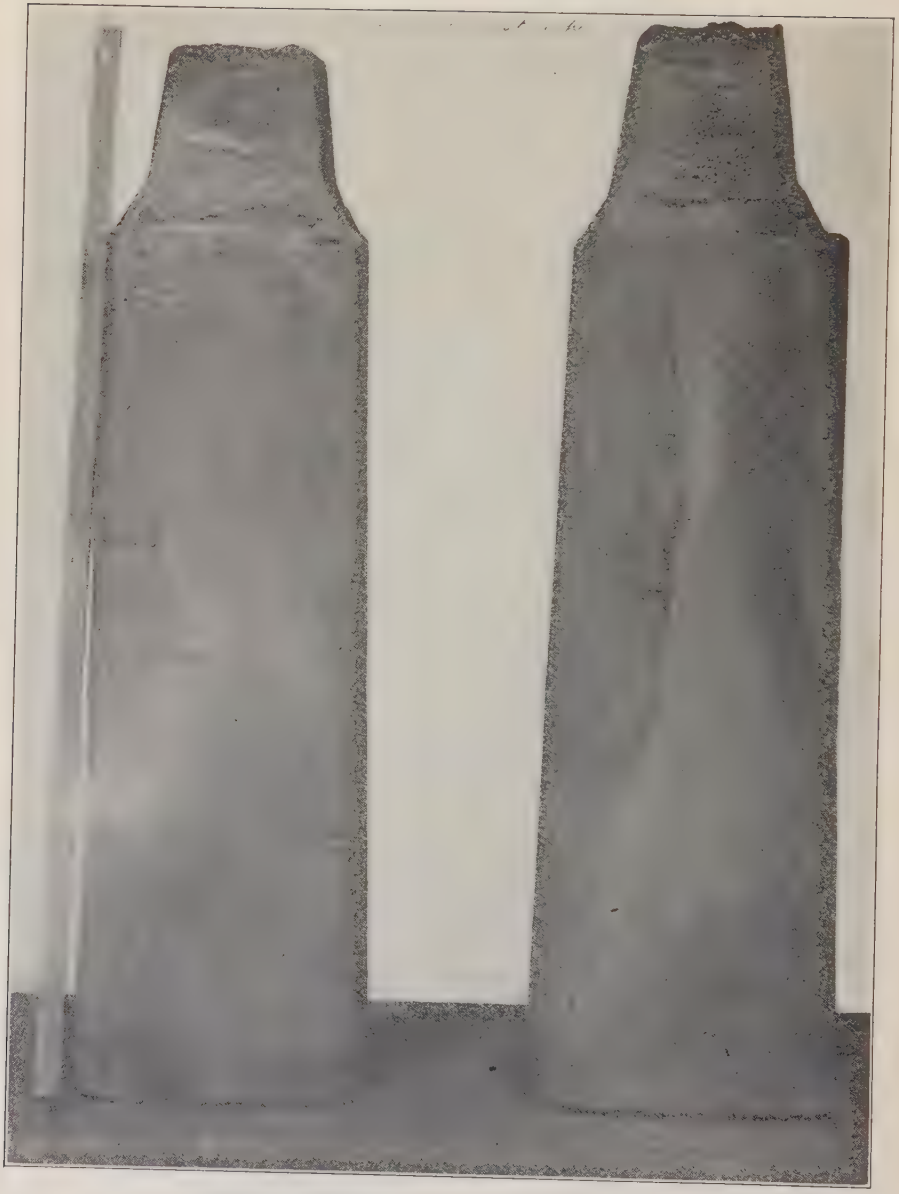


PLATE VIII.—ANOTHER VIEW OF THE INGOTS SHOWN IN PLATE VII.

any government inspector the writer has ever met, yet in a rail, upon which human life so largely depends, it seems to be taken as a matter of course that such a product may safely pass into service.

As regards steel of piping nature, but cast without a feeding head, such as shown by *B* in Plate V, this is probably the more dangerous of the two types shown in this plate, as in the case of a warm heat the piping will descend to a greater depth. While a certain portion of this may be got rid of in the discard, this often amounting to a large percentage, yet this means great waste. Moreover, and this is still more important to be borne in mind by the user of the rail, there are cases showing that below the portion where the pipe has been supposed to terminate, this sometimes breaks out again and is not detected. In other words, a discard may be taken and show sound material, yet below this it is easily possible to have further dangerous segregation and piping.

As regards the length of time required for ingots made under the system now described to cool down before they can be dealt with, it is estimated that probably not more than about 15 min. would be necessary.

To further differentiate between the various types of ingots as now produced and as made by the writer's system, the reader is referred to Plate IX, which gives this in a graphic manner.

Fig. 1 represents badly rising steel, which is full of blowholes almost entirely throughout the mass; the steel rises and shows practically no piping. Although in rolling this steel the blowholes are apparently closed up, without doubt by a careful examination of its microstructure it would be found that the material can never be as strong or as dense as steel made sound in the first instance. The product is, moreover, liable to segregation in the upper portion. A large discard is necessary, in some cases as much as 20 per cent. or more.

Fig. 2 represents steel which neither rises nor settles. In this steel there are fewer blowholes, but the steel is still unsound, with a tendency to pipe. This steel contains considerable unsoundness, and owing to the tendency to pipe, troubles are liable to occur with segregation and piping. Probably at least 15 to 20 per cent. discard is necessary.

Fig. 3 represents settling steel which is not fed in the upper portion, as in the case of those ingots made by the writer's system. This pipe sometimes runs half the length of the ingot. Moreover, such material has sometimes a tendency to appear sound, then below this to break out again into piping. While this steel is free from blowholes, material of uncertain nature is obtained, owing to its badly piped condition. It is probably this kind of steel which gives the dangerous rails below those made from what are known as the upper or *A* and *B* portions of the ingot. In other words, it is quite possible piped or unsound *C* rails may be obtained from such ingots.

Fig. 4 represents an ingot made under the writer's system. The steel

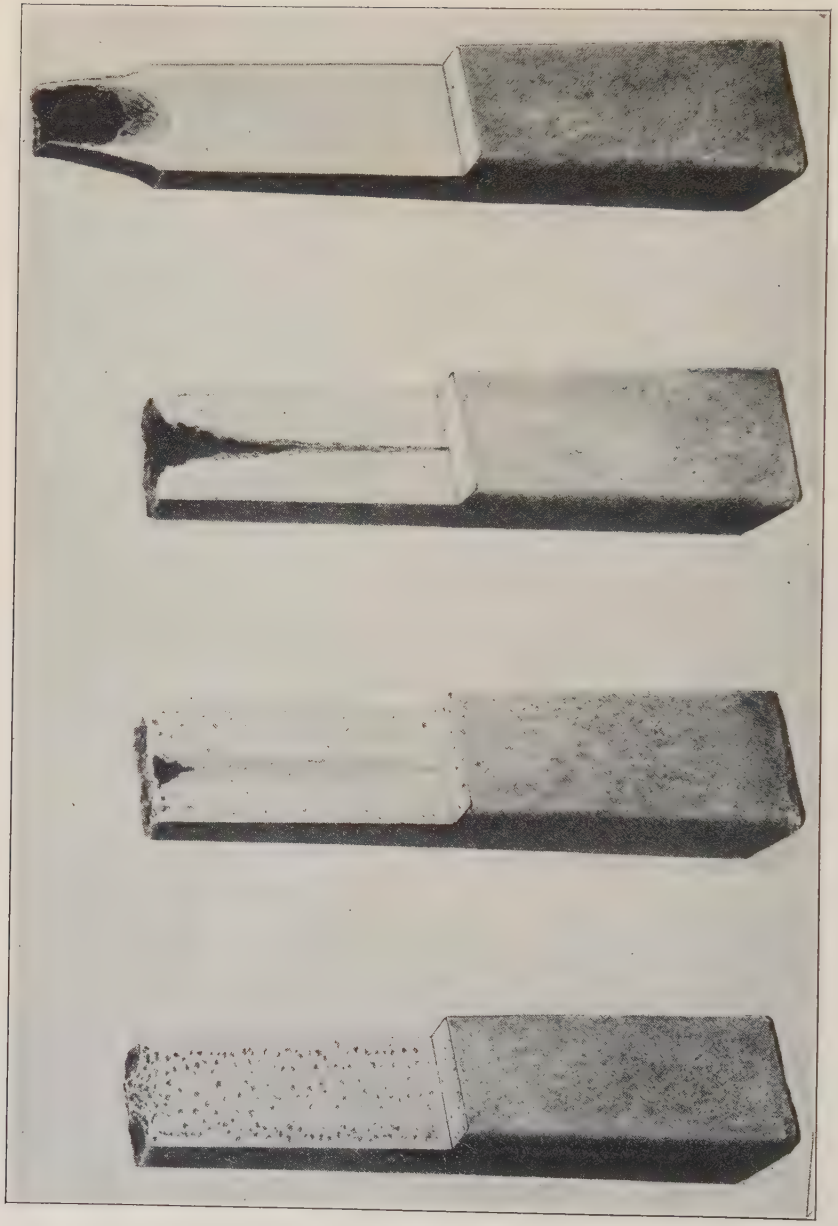


Fig. 1. Fig. 2. Fig. 3. Fig. 4.

PLATE IX.

used in the first place is perfectly sound and free from blowholes. The feeding arrangement enables from 88 to 90 per cent. of the total weight of ingot to be safely used. In some cases, no less than 93 per cent. of sound billets have been obtained from ingots made in this manner; that is, 93 per cent. of the material showed neither blowholes, piping, segregation, nor other defects.

With reference to the cavities or hollow portions in the heads of the ingots made by the writer's system, in view of the extraordinarily large nature of these cavities, the writer decided to make the following experiment with a view to determine the exact value of the settling of the steel from the head above the ingot into the ingot itself. He believes that such a method of testing ingots has not before been carried out. It will be understood this has only been made for experimental purposes, because by an examination from the top of the ingot, occupying only a few seconds, it can be easily determined whether the ingots when produced in course of manufacture, whether in small or large quantities, possess the necessary soundness. If the steel was not sound there would be no cavity, so that the quality of the steel is self-apparent.

Nine 15-in. ingots were taken (weighing about 3,600 lb. each) as they came through, each of which had the sand head and the writer's improved method of feeding carried out on them. After the ingots had cooled down, the hollows or cavities in the sand heads were filled with water, then the water poured out and carefully measured. Table I shows the results obtained.

TABLE I

Ingot Number	Cubic Inches	Pounds of Steel	Percentage of Weight of Ingot.
1	457	128	3.57
2	549	154	4.30
3	457	128	3.57
4	457	128	3.57
5	472	132	3.68
6	488	137	3.82
7	518	145	4.05
8	579	162	4.52
9	488	137	3.82
Average.....	496	139	3.88
Maximum.....	579	162	4.52
Minimum.....	457	128	3.57
	Weight of ingot, 3,600 lb.		

The average weight for the nine 15-in. ingots showed that 139 lb.,

with a minimum of 128 lb. and a maximum of 162 lb., passed from the head portion into the ingot itself. This percentage is represented by an average figure of 3.88 per cent. In other words, about 4 per cent., or 140 lb., of the total weight of the ingot or ingots cast passed from the upper or feeding head into the body of the ingot.

There could not be a more striking illustration of the quality of the ingots produced and of the value of this system. Even in the case of the minimum percentage weight of steel which passed from the head of the ingot itself, there is a weight of no less than 128 lb. of steel short in the head, and therefore present in the ingot itself, all adding to the soundness and proper feeding of the piping which would otherwise occur.

To show still more clearly the important information obtained from these experiments, let it be assumed that the cubic capacity of each of the 15-in. ingots in question was approximately 12,500 cu. in. Therefore, but for this feeding, there would be a general want of solidity, chiefly at the upper portion of the ingot, to the extent of say 500 cu. in., say 4 per cent. of the whole capacity. It is surely readily apparent why an ingot which is not fed must perforce be deficient in homogeneity.

It is not claimed that ingots made in the ordinary manner are deficient to the full extent of say 4 per cent. There is a feeding effect from the steel in the upper portion of the ingot, but it cannot be done efficiently, as the steel quickly freezes on the outside of the mold and on the surface of the liquid steel exposed to the air. Moreover, there is always an uncertainty as to how good or how bad is the resulting material. If the steel is piping very much, the trouble will be worse than when it is piping less. In any case, as the steel solidifies in an ingot of this size, the natural law of contraction demands that about 500 cu. in. has to be dealt with. The writer cannot see how this can be efficiently and cheaply met except by some such method as described in this paper.

While the results necessarily vary slightly, because the sizes of the head portion nearest the top of the mold formed in sand are not always uniform in length, as the steel shrinks down slightly more on the outside in some cases than others, on the whole the maximum and minimum figures of $4\frac{1}{2}$ and $3\frac{1}{2}$ per cent. of the total weight of the ingot having passed from the head into the ingot itself, show very uniform working; if the heads were absolutely the same depth in each case, there would be practically no difference.

Without wishing to exaggerate, it is easy to picture to oneself what would be the character of each of these ingots if not made and treated under the method now described. The piping would have probably run down the ingot itself, requiring a discard of probably 25 to 33 per cent.

Although water cannot be poured into the cavity of a red-hot ingot, yet the cavity can be determined in each ingot by a cursory examination

while at a red or yellow heat, involving only a few seconds of time. It will therefore be seen that every ingot can be readily checked by such cursory examination.

While in ingots made in the ordinary way as above mentioned a certain amount of the fluid steel passes from the upper portion to the lower, still in so doing it is robbing the quality of the upper portion of the ingot itself, which has no fluid metal above it to feed or take the place and supply the deficiency thus created. It will readily be understood therefore why the upper portion of ingots is so seriously affected as regards their soundness, also why segregation occurs. The occurrence of these defects varies according to the type of steel, whether rising, semi-rising, or settling nature, as shown by Plate IX.

Again referring to this question of the cavities, if, as proved by these experiments, in the ingots made under the writer's system the metal in the sand head portion in descending has without doubt filled or prevented the formation of what would otherwise have been unsoundness, piping, loose structure, or segregated material, in unfed ingots made in the ordinary way there must be steel of loose structure; if not, then in many cases absolute unsoundness and segregation.

Moreover, and this is a most important point, the steel in the "fed" ingots being maintained fluid in the head portion, continues to exert its ferro-static pressure, whereas with ingots made in the ordinary way the ferro-static pressure on the center portion of the ingot is so slight that it produces very little beneficial effect. Further, without the feeding head above the ingot proper, the outside of the ingot in the ordinary ingot mold becomes rapidly chilled and frozen, so that it cannot contribute its proper share to the feeding of the remaining portion of the ingot. It is not therefore to be wondered at that rails rolled from the *A* and *B* portions of an ingot made in the ordinary way are liable to unsoundness or piping or both, and are also often full of impure segregated material.

There would probably be more dangerous ingots but for the fact that the steel maker tries to avoid this type of steel, and aims to make steel which when poured into the ingot will not pipe. Nevertheless, he is still fighting against a natural law. If piping steel is checked or avoided, he runs the risk of producing unsound steel, especially in the upper portion of the ingot, more or less permeated with blowholes. Thus, owing to lack of feeding from the upper portion, the center, or that portion on the axis line of the ingot, must be of inferior nature, as the piping characteristics persist for quite a long way down the ingot. This, as before mentioned, is for the reason that owing to want of ferro-static pressure, the ingot lacks feeding from above, which in the system of casting ingots now described is maintained to a very late stage; that is, until or close upon actual solidification takes place. There is always fluid steel in the upper portion of the ingot to feed the piping and shrink-

age, both of which must occur, as they follow a natural law. This, too, is the reason why there is so little segregation in ingots made under the writer's system, and also explains why the ferro-static pressure is kept up to a very late stage. In fact, check or hinder ferro-static pressure, and segregation with its bad effects at once commences. In the case of "fed" ingots, the smaller amount of segregation which occurs takes place outside the ingot proper, that is in the head. This is well illustrated by the results shown in Figs. 4 and 5 of Plate I.

With steel of piping nature poured into the ingot molds and not fed, it need cause no astonishment to find that rails, even from *C* and *D* portions of the ingots, may be of material with loose structure, consequently weak if not actually unsound, thus giving inferior or bad results in service. This is shown by Fig. 3 in Plate I, which shows that with piping steel not properly fed, there is extreme danger of this pipe extending a long way down the ingot; in fact, in some cases it has been found to go as much as two-thirds the length of the ingot. Ingots have also been found in which the piping has apparently stopped, only to be resumed below the sounder steel.

The experiment carried out by the writer some years ago, by the pouring of copper into the upper portion of an ingot 15 or 20 min. after casting, showed how serious is this want of ferro-static pressure in the material situated on or near the center or axis line of the ingot in ingots which have not been properly fed. The copper finds its way down to the bottom of the ingot, although added 15 min. after casting. In any case, if there is no definite pipe at the bottom portion of such ingot, there is still material of loose or open structure, which means weak steel. Although this may not be apparent by fracture to the naked eye, nevertheless it exists and can generally be detected by an examination of the microstructure. In other words, notwithstanding that the product to be used may come from the lower half of the ingot, yet in unfed ingots it will be weak and not able to stand severe stresses. This is probably the real explanation of the serious breakages which sometimes occur in even *C* and *D* rails. Imperfect material is present; only time and sufficient working stresses are wanted to develop its existence and weakness.

It is true that some portions of the cavities in ingots have been measured, but probably not in the manner described by the writer.

Although in the examination of the top of an ingot cast in the ordinary manner and from steel which "settles," there is external evidence of some piping, this is irregular and varies considerably. Therefore, in the "best" ordinary ingot proper evidence is slight as to how much or how little the steel has piped. Dr. Dudley has pointed out in his interesting paper to the Institute on Piping and Segregation of Ingots of Steel and Ductility-Tests for Open-Hearth Steel Rails, read in February, 1913, that in such ingots the piping is divided into two kinds, the upper, or what

may be termed the visible pipe, and the lower, or hidden pipe, the extent and character of which can only be determined by cutting open the ingot. In the ingots cast under the writer's system, *all the cavity or pipe is open* and can readily be inspected from the top; its extent can be determined whether in the hot or cold condition. It is therefore not necessary to cut open the ingot. In other words, such cavity produced by the piping is "the outward and visible sign of an inward and spiritual grace"—"spiritual grace" in this case meaning "soundness."

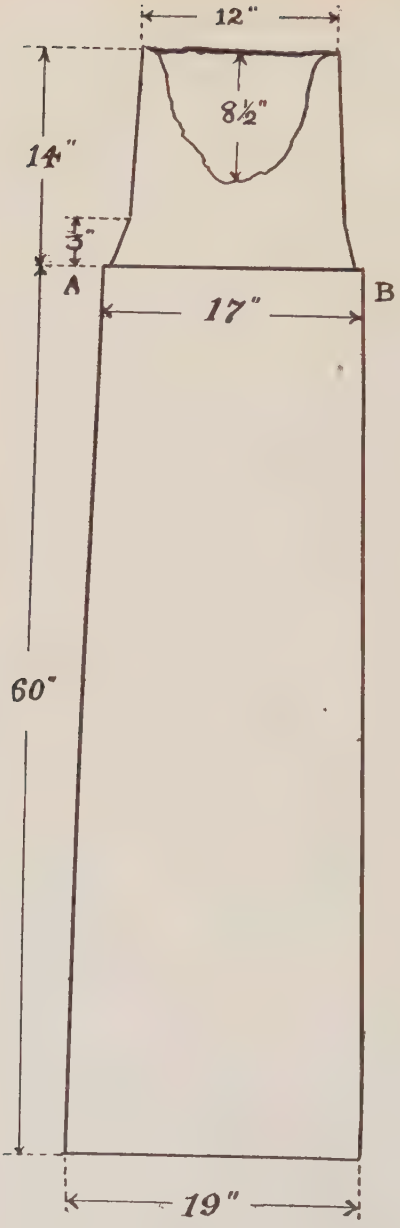
Figs. 1 and 2, Plate X, further explain this.

Fig. 1 represents an 18-in. ingot made on the writer's system, and as shown in Plate VII already referred to. *All the cavity produced by the piping* is easily visible and measurable from the top. If the ingot is allowed to go cold, the cavity can be measured and its cubic contents ascertained by filling such cavity with water, then pouring this out and taking its volume. If desired, the same object could probably be accomplished while the ingot was hot, in that case using a metal of suitable melting point. Such an ingot possesses no upper or lower cavity, it is all in one. The full characteristics of the ingot in this respect, and whether hot or cold, are known and determined by such cavity.

Fig. 2 shows the section of an ordinary 21-in. ingot made from steel which is of piping nature. It will be seen that there are two separate cavities, an upper and a lower one. While the former can be measured, the information obtained would not be of much practical value. As regards the latter, it is, so to speak, covered by a metallic diaphragm and cannot be measured except by cutting or machining open the ingot when cold. Moreover, it will be seen that the center portion of the ingot, that is the metal over and surrounding the central vertical or axis line of the ingot, is not sound for quite a long way down. Below where it appears to be quite sound to the eye there are still segregation results to be dealt with. In other words, a considerable portion of this ingot must be "discarded" or "cropped," either in the ingot itself or the rail produced therefrom, before a sound, safe rail can be obtained, that is, one which is perfectly sound not merely to the naked eye when examining a fracture of the ingot or the rail from such ingot, but is so when its microstructure is examined, or sulphur prints or etchings are taken.

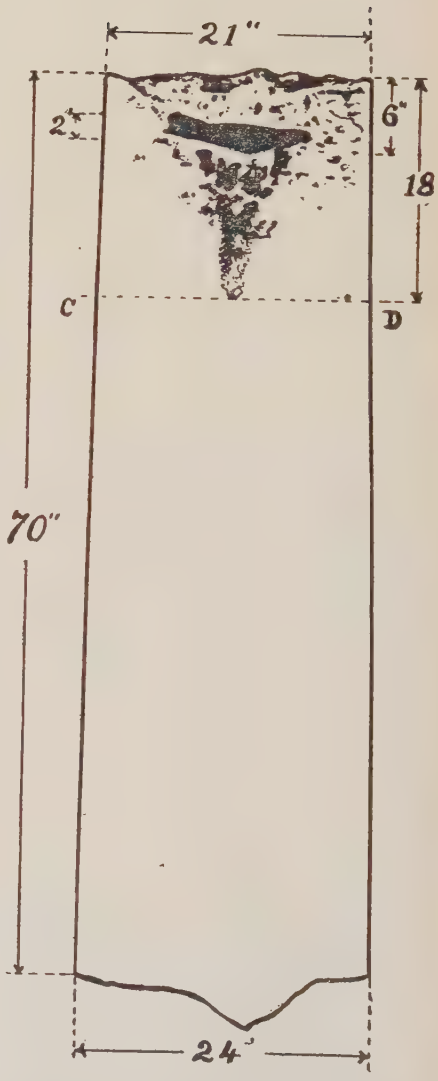
This real and true soundness is obtained in ingots made under the writer's system; that is, almost immediately under the cavity, and after a discard or crop has been taken of even as low as 7 per cent. in some cases, and certainly with 8 to 10 per cent. Therefore, such products as billets, bars, rails, sheets, etc., produced therefrom are practically sound at any portion of their mass.

This would appear to be a considerable advance as compared with ordinary practice, and even as compared with ordinary ingots of a char-



In the case of this ingot, all the metal is sound and free from piping and segregation below the line *A–B*.

Fig. 1.—Length of Sound Ingot Proper, 60 In.



While the metal is sound below the line *C–D*, there are still segregation effects probably several inches further down.

Fig. 2.—Length of Sound Ingot Proper, 52 In.

Plate X.

acter such as above referred to, which in themselves are certainly superior to many ordinary ingots made.

The ingot in Fig. 1, representing an ingot made under the writer's system, shows no unsound portion, that is to say, blowholes are entirely absent. About $8\frac{1}{2}$ in. from the top of such ingot, *segregation has disappeared.* Similar comparison with Fig. 2, representing what may be termed a high-class ordinary ingot, shows that at a point no less than 18 in. from the top there is still unsoundness. Moreover, below even the sound portion, there is segregated material which must be discarded before safe and sound structural steel can be obtained. It will be seen the discard of the former must be far lower, yet better and more sound material is produced.

The writer is aware that the above comparison is made between an ingot prepared under his system and an ordinary ingot which has been allowed to become cold. As Dr. Dudley has pointed out, there is considerable improvement in such ingots by handling them quickly while very hot. There is no doubt this is quite correct; yet in ingots cast under the writer's system, which can also be handled while very hot, it is difficult to see how an improvement can be effected in ingots which are already practically perfect.

At any rate, there certainly could not be a reduction in quality. In other words, if there was any difference in ingots made under the writer's system, the difference between "hot and cold" ingots would be in a favorable direction. The writer, however, thinks that these differences, that is, between the same ingots hot and cold, whether as regards the ordinary or the ingots made under his system, can hardly be very considerable, for after molten steel has once passed into the "solidus" as compared with the "liquidus" condition, there can surely only then be the ordinary contraction changes, or what may be termed "linear" contractions. These do not materially affect the volume of the cavity or the unsoundness, which if they exist remain practically as they are when the steel congeals into the "solidus" condition.

In view of the greatly increased service now met with on railroads in nearly all countries, resulting from heavier and more trains, higher speeds, whether freight or passenger, it will be seen that, in the nature of things, rails and similar products must be made from sound steel and from ingots each of which must be known to be as perfect as human ingenuity can accomplish. If this and other papers on the subject help to bring us nearer the goal and also help to remove what would appear to be a slur upon the metallurgist, then the labor involved in the many researches being carried out in this country and elsewhere will be well repaid.

Rolled Steel Roll Shells

BY JAMES C. H. FERGUSON, SAN FRANCISCO, CAL.

(Pittsburgh Meeting, October, 1914)

THE fact that little if anything has appeared in the technical press or in the *Transactions* of the Institute on the subject of roll shells proper, used in various grinding appliances such as Cornish rolls or Chilean mills, may be taken as an incentive, if not an excuse, for the present article. No attempt is made to enter into a discussion as to the theory or mathematics of crushing as performed by rolls, the subject being treated in a general way with respect to the manufacture of roll shells and some salient points touching on their adaptability and wear in service.

While it stands to reason that skill coupled with judgment in the mechanical design of rolls, Chilean mills or other types of grinding machinery, with special reference to strength of parts, nicety of adjustment and other mechanical features, stand for efficiency in their operation and satisfaction in service, the exceedingly important item of roll shells or other wearing parts entering into their construction should not be lost sight of. In other words, a well-designed piece of grinding machinery is just as inefficient in service, when equipped with poor roll shells or shells made of the wrong or an inferior grade of metal, as good roll shells made according to the best practice the steel maker knows, working on an old or antiquated set of rolls.

Unfortunately there is a tendency occasionally for the machinery builder to blame the steel maker or *vice versa* if roll shells do not wear well or evenly, or if they break before being worn down to their ultimate thin point. There is no occasion for this attitude, as the product of each must stand on its own merits. But it is essential that the two work in harmony since the machinery builder usually does not furnish the shells for his rolls or grinding mills, but depends on the steel maker, who makes a specialty of this rolled product, to give the particular needed element of efficiency to his machine.

HISTORY

Without going into an elaborate history of the subject, the following may be mentioned treating on the evolution of roll shells. During the early days of mining-machinery construction, cast iron was used

for practically all such grinding devices as those under consideration, usually some good mixture of gray iron being the most suitable for wearing qualities. Sometimes, though not often, shells or muller rings, chilled to a depth of from ½ to 1 in. through the face, were used.

But it soon became apparent that cast iron, no matter how well selected as to grade or mixture, or how carefully molded or chilled as to physical appearance, was not a suitable material to use for grinding rings or dies, since, for the proper performance of their work, absolute homogeneity and freedom from blow-holes or other imperfections are highly important. Steel, that is cast steel, therefore, was next and most logically considered as being more suitable to fulfill the *desiderata* just mentioned. Although it was more expensive than cast iron, it was harder and tougher, and of infinitely better wearing quality. But even cast steel failed to give satisfaction particularly on the high-speed rolls of narrow face and large diameters designed by the leading manufacturers during the past two decades. It was for that reason that the steel maker soon recognized the importance of meeting the situation and has during the period mentioned evolved the rolled-steel roll shell. This was a natural evolution, and during the past 10 years practically no other material has been used except for some small or slow-running mills where the expense of steel shells was not warranted.

Therefore, it is with steel roll shells only that the steel maker is concerned when he furnishes either the machinery builder or the mill operator these wearing parts, and for a better understanding of the subject a few words regarding their chemical and physical requirements may not be amiss, before entering upon a consideration of their manufacture.

Chemical and Physical Requirements

While as mentioned above the application of steel shells as wearing parts in the various types of grinding mills was comparatively new, the purely mechanical features of their manufacture was easily accomplished by the use of tire-rolling mills which for many years previous had been used for making locomotive driving-wheel and car-wheel tires. The mechanical manipulation being practically identical, it only required that the steel maker select the proper grade and composition of steel for the roll shell, not too hard, for danger of breaking, nor too soft, for fear of wearing out too fast. Of course it required much experimenting and many years' experience to attain the desired results.

With reference to the average chemical and physical properties entering into the manufacture of roll shells, the following may be given as forming the best practice by American steel makers:

Chemical Properties	Per cent.
Carbon	0.65 to 0.80
Phosphorus, not to exceed	0.05
Sulphur, not to exceed	0.04
Manganese	0.60 to 0.85
Silicon	0.15 to 0.30

The above are the average ranges used. Shells which have given good satisfaction in actual operation had the following analyses in the heats from which they were produced:

	Per cent.	Per cent.	Per cent.
Carbon	0.70	0.73	0.78
Phosphorus	0.035	0.029	0.015
Sulphur	0.022	0.018	0.028
Manganese	0.741	0.786	0.745
Silicon	0.253	0.252	0.258

These results show that the shells run very uniform in composition.

Physical Properties	
Tensile strength, pounds	125,000
Elastic limit, pounds	72,000
Extension, per cent	10
Contraction of area, per cent	15

The above physical properties are given as being of general interest but they should not be taken as having any particular bearing on the successful working of roll shells, as they do not in any way exhibit the qualities of the metal in its resistance to abrasive wear. After all, what is desired, as previously stated, is a shell of a metal hard enough so as not to wear out too quickly and yet not so hard as to break before wearing out.

Manufacture

Briefly, the process of manufacture of roll shells is this: A long cylindrical steel ingot is cast by pouring molten metal into a cast-iron mold as illustrated in Fig. 1. These ingots are about 7 ft. high and the circumferential measurement of the ingot varies according to the size of the roll shell desired. When the metal is cold the molds are stripped from the ingots, which are then sliced cold in a lathe into a number of sections called billets (Figs. 2 and 3), each piece resembling a large cheese. Sometimes the ingots are sliced or cut into billets while hot, but of course cold slicing is a vastly superior method as it gives an opportunity to make a thorough examination of the center of the billet.

The top portion of these ingots, in which the segregation of impurities predominates, is discarded, and only those portions used which are free from impurities. It is a well-known fact that there is a tendency in all castings while cooling for slaggy materials and impurities to segregate and form toward the center, where the shrinkage produces a pipe or axial

opening. The aim, therefore, in the manufacture of good roll shells must be to retain only those portions of the ingot which are on or near the outer circumference, which having cooled the most rapidly necessarily have the most uniform fiber texture and are the freest from impurities.

In order to illustrate the practice of slicing these ingots with regard to discarding the top section, reference is made to Fig. 4, which illustrates a segregation of impurities and defects due to the shrinkage of the metal and shows that when cooling, these impurities concentrate at the center and near the top of the ingot. It conclusively shows the necessity of making

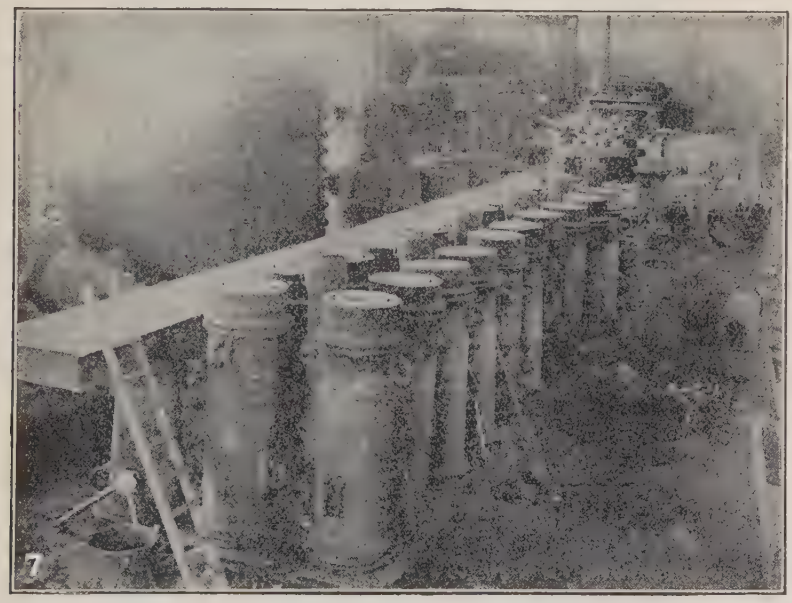


FIG. 1.—MOLDS FOR CASTING STEEL INGOTS FOR ROLL-SHELL MANUFACTURE.

a proper discard from any steel castings, in order to insure against troubles due to impurities.

These billets after being sliced, although not of the proper dimensions, actually represent the weight of the finished product desired, plus the proper allowance for loss in heating, forging and rolling, and for machining afterward if the shell calls for machine work. The billet is heated and forged out roughly and flattened under a steam hammer to the approximate diameter and face desired. A hole is punched through the center in the same operation and then, at the same heat, "beaked" on the horn of the anvil; this means hammering the rough ring thrown over the horn, producing an increase of the outside and inside diameters, and giving the ring the proper ratio, roughly, of diameter to face required.

These rough rings, after being thus forged, are again placed in a heating furnace and the temperature raised to the proper degree for rolling.

A modification of this hammering process of the billets, is used by one of the manufacturers who has the most recent and up-to-date plant, and who, after heating the billet to the required temperature, works it down

FIGS. 2 AND 3.—STEEL INGOTS CUT INTO BILLETS.

under a 5,000-ton hydraulic press, which gives the metal a good reduction. During this pressing, which is done in three operations, the billet is flattened, a hole is punched in the center, and the rough ring prepared for the rolls, similarly, but unquestionably more thoroughly, than by hammering the billet.

The rolling out of the shells to the desired size is done on a tire mill, where they are revolved through pressure rollers, either in a horizontal or vertical plane. During this operation all faces of the shell are subjected

to a very high hydraulic roll pressure, which insures a thorough reduction of the metal, giving it the necessary work to develop an ideal structure for the severe service to which roll shells are usually subjected in actual use. In this rolling the inside and outside faces and the two sides of the shell are engaged at the same time, thereby increasing the diameter of the shell by squeezing out and lengthening its circumference until the desired size is obtained, both as regards diameter and face.

As stated before, this rolling is done on the same mills used for rolling locomotive driving-wheel tires or car-wheel tires. This method has now

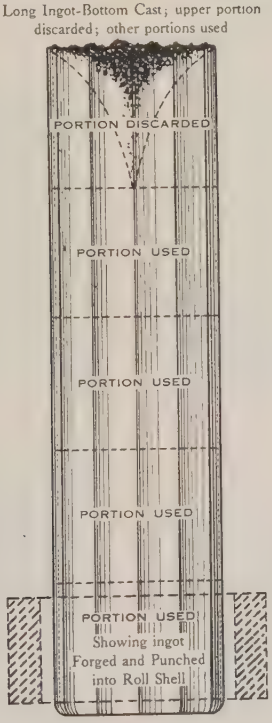


FIG. 4.—DIAGRAM SHOWING RELATION OF FINISHED ROLL SHELL WITH REFERENCE TO CENTER LINE OF INGOT.

reached a very high degree of efficiency and in a number of cases shells are put into service just as they come from these rolls, the outside diameter, together with such tolerances as may be permissible, being so close as to require no further machining on the face. This leaves a hard skin due to the quicker chilling of the metal on the periphery, which is very useful in grinding and enhances the life of the shell considerably. The inside of the shell is left rough if it is to be secured to the center of the rolls by wooden wedges for wet grinding. If the bore of the ring is to be machined either straight or tapering, proper allowance is left for that

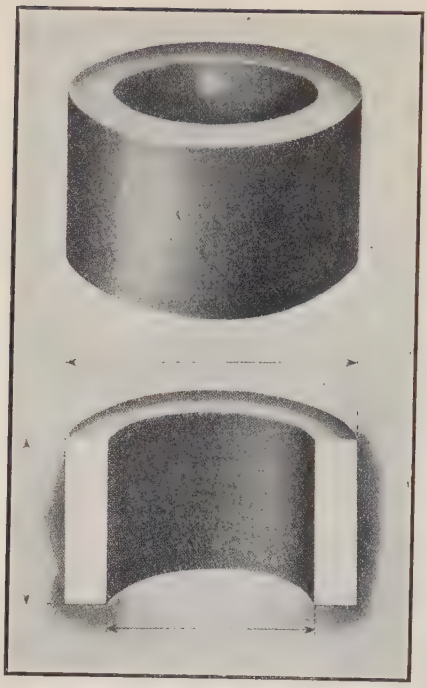


FIG. 5.—STRAIGHT BORE.

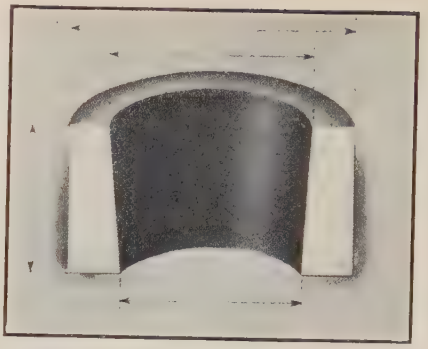


FIG. 6.—SINGLE-TAPER BORE.

FIG. 7.—DOUBLE-TAPER BORE.

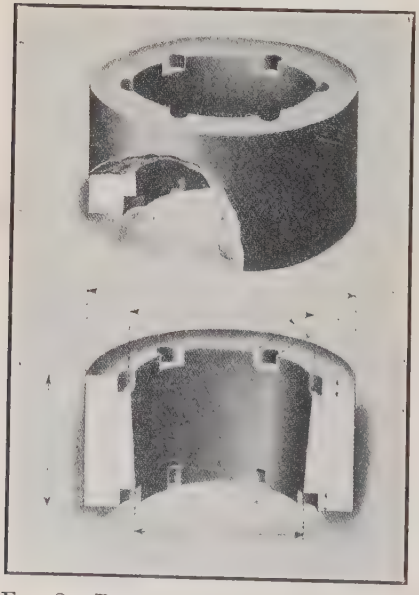


FIG. 8.—RECESSED FOR DRAW BOLTS.

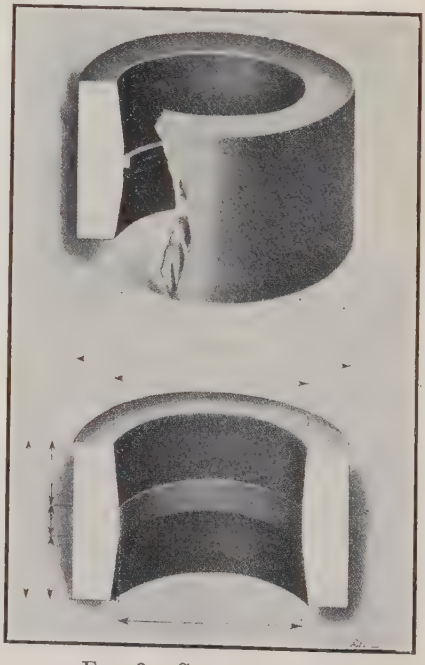


FIG. 9.—SPECIAL BORE.

FIGS. 5 TO 9.—PREVAILING TYPES OF BORES OF ROLL SHELLS.

purpose on the inside, but the outside or wearing face of ring, being finished practically smooth by the rolling operations, should preferably be left in that condition, by reason of the advantage mentioned above.

Figs. 5 to 9 illustrate some of the prevailing types of bores of shells as used in rolls manufactured by the principal mining-machinery builders in the United States and are usually designated as follows: Fig. 5, Straight bore; Fig. 6, Single-taper bore; Fig. 7, Double-taper bore; Fig. 8, Recessed for draw bolts; Fig. 9, Special bore.

With regard to the rolling of shells, there would seem to be a limit of efficiency governed by the width and thickness of shells. Manifestly a mass of metal thicker than 5 in. cannot be sufficiently worked by the pressure of rolls, unless the rolls can be made very much more powerful than those in use at present, particularly when the combination of rolling and pressing naturally has its relative limits. Then again, the width, regardless of thickness, would have its limit at about 15 or 16 in., because a very wide mass, even thinner than 5 in., would experience the same difficulty in being thoroughly worked. It may therefore be assumed that shells over 5 in. thickness and wider than 16 in. cannot be rolled advantageously in a tire mill and produce homogeneous and dense metal, which will give good service and wear.

The following gives the limits of sizes for rolling as used in the practice of one of the largest tire manufacturers in the United States: Roll shells 8 to 10 in. wide, 108 in. outside diameter by 3 in. thick; over 10 to 16 in., or under, 68 in. outside diameter by 3 in. thick.

Shells over 16 in. wide and thicker than 5 in. and weighing say 5,000 lb., or over, are produced to the best advantage under a hydraulic forging press of adequate power, as a strictly forged, instead of a rolled product. Sufficient stress cannot be laid upon the necessity of thoroughly working the metal in these forgings all the way to the center by the use of machinery of this kind, which is sufficiently heavy and powerful to perform the necessary forging operation and is better able to do so than rolls. The writer is convinced that most of the cases of uneven wear in roll shells of extreme widths, where not due to improper setting or improper mill feeding, or both, is largely owing to the fact that the metal is not of uniform density, resulting from being worked in a tire mill, which was too light for the heavy and thick section passing through the rolls.

Life of Shells

The limits of this article do not permit as exhaustive a treatment of this sub-heading as the writer would like to submit, based on data gathered during the past 13 years in furnishing the mining public in the West with steel roll shells and material of kindred nature, but for the sake of generality as well as brevity the following examples taken from users of roll shells in the widely separated States of Montana, Colorado,

Sonora, Arizona and Utah will illustrate what may be considered good average records of the life of steel roll shells and Chilean mill wearing parts.

Mill No. 1 (*Montana*).—The mills used were 6-ft. Evans-Waddell and 6-ft. Monadnock type, being fed by revolving distributors in the center. The ore crushed was very hard with about 65 per cent. silica. The size of feed was 3 mm. and discharge, 16 mm. The rate of crushing was 80 tons per 24 hr. or 2,400 tons per month. The life of the tires was 4 months per set of three, running continuously; the life of dies, 2 months, running continuously. The amount of rock crushed per pound of steel consumed was 1.06 tons.

Mill No. 2 (*Colorado*).—At this plant, 6-ft. Evans-Waddell mills, fed by revolving distributors at the center, were used. Roasted ore, assaying 60 per cent. silica, was fed, sized as follows: On 5/32, 4 per cent.; on 10 mesh, 15; on 16 mesh, 16; on 20 mesh, 7; on 30 mesh, 11; and through 30 mesh, 47 per cent. The discharge gave the following screen analysis: On 30 mesh, 1 per cent.; on 40 mesh, 13; on 60 mesh, 20; on 80 mesh, 12; on 100 mesh, 17; on 150 mesh, 7; and through 150 mesh 30 per cent. Crushing was at the rate of 173 tons per 24 hr. The average life of tires and dies determined by life of dies was: Steel maker No. 1, 183.3 days; steel maker No. 2, 164.2 days. The average tonnage crushed per set: Steel maker No. 1, 31,711 tons; steel maker No. 2, 28,407 tons. The average weight of shell, 1,488, set (3), 4,464 lb.; of die, 3,029 lb.; total, 7,493 lb. The number of tons crushed per pound of steel consumed was: No. 1, 4.25 and No. 2, 3.8, which gives the number of pounds of steel consumed per ton of ore crushed as follows: No. 1, 0.236 and No. 2, 0.264 lb. The average weight of scrap per set was 1,640 lb.

Mill No. 3 (*Sonora*)

Dept.	No. of Pieces	Diameter, Inches	Face, Inches	Weight per Set, Pounds	Life, Days	Tons Ore Crushed	Pounds Steel per Ton Ore	Size Feed	Size Discharge	
A	10	36	16	13,615	270	308,000	0.044	1½ in.	1 in.	Roll shells
B	4	36	16	5,444	135	81,000	0.067	1 in.	⅜ in.	Roll shells
	8	27	14	7,256	80	118,400	0.061	5/16 in.	2 mm.	Roll shells
	3	54	7	7,892	182	35,100	0.224	.....	.....	Tires and die 6 ft. mill.
	2	70	7							
	5	60 and 44	7	6,502	425	21,625	0.304	¼ in.	1½ mm.	Tires and die 5 ft. mill.
C	4	36	16	5,444	195	25,798	0.210	1 in.	⅜ in.	Roll shells
	2	60	7	6,502	240	69,960	0.093	¼ in.	.....	Die rings, Bryan mill tires.
	3	44	7							

Mill No. 4 (Arizona)

Data on shells from two sets of 54 by 24 in. rolls (dry crushing). One shell, 54 by 24 in. (in two pieces. 54 by 12 in.).

Outside diameter, inches	54
Inside diameter, inches	$43\frac{5}{8}$
Original thickness, inches	$5\frac{1}{16}$
Ultimate thickness, inches	1
Wear, inches	$4\frac{1}{16}$
Original weight (two pieces), pounds	5,408
Scrap weight (two pieces), pounds	1,130
Loss, pounds	4,278
Tons through crusher	1,266,000
Tons crushed each set of rolls	538,000
(Feed, 3 in. size; product, 1 in. size.)	
Loss of steel per ton crushed, pounds	0.0159

Note.—These shells were never taken out of the machines since started. They were not entirely worn out but had been in use $2\frac{1}{2}$ years. During that time they were trued up occasionally in the machine by attaching a lathe tool holder to the main frame and then turning the rolls backward by a back-geared electric outfit through the medium of the regular roll belts.

During the period that the two sets of rolls were in use they crushed 1,076,000 tons of ore from 3 in. size, which had been produced by a No. 8 McCully crusher (with the fines screened out *en route* to the rolls), to a product approximating 1 in. size. Approximately 25 per cent. of the crusher product was taken out by the screens ahead of the rolls.

Mill No. 5 (Utah)

Record of Roll Shells in use at Sample Mill, June, 1914

Section Number	Roll Size Inches	Thickness of Shell, Inches	Date Installed	Date Replaced	Period of Service, Mo. Dys.	Estimated Tonnage	Size of Crushing, Inches	Cause for Change
1	42 by 15	5	8–1–12	2– 9–13	5 8	54,000	$\frac{1}{2}$	Surface corrugated
1	42 by 15	$3\frac{3}{4}$	2– 9–13	6– 8–13	4	38,000	$\frac{1}{2}$	Surface corrugated
1	42 by 15	$2\frac{1}{2}$	6– 8–13	11– 6–13	5	36,000	$\frac{1}{2}$	Surface corrugated
1	42 by 15	$3\frac{3}{4}$	11– 6–13	4–27–14	5 21	50,000	$\frac{1}{2}$	Worn out
1	54 by 24	5	8–28–13	Still on	.....	95,000	$\frac{3}{4}$	
2	42 by 15	$3\frac{3}{4}$	2– 3–13	5– 1–13	3	27,000	$\frac{1}{2}$	Surface corrugated
2	42 by 15	5	5– 1–13	9–27–13	4 26	40,000	$\frac{1}{2}$	Surface corrugated
2	42 by 15	$3\frac{3}{4}$	9–27–13	12–26–13	3	24,000	$\frac{1}{2}$	Slipped off.
2	42 by 15	$3\frac{3}{4}$	12–26–13	Still on	.....	58,000	$\frac{1}{2}$	.....
2	54 by 24	5	6–28–13	Still on	.....	116,000	$\frac{3}{4}$	.....

Material in 42 by 15 in. shells is rolled tire steel, the 54 by 24 in. shells are forged throughout.

Four sets of rolls are provided for two machines, and as a different set of shells is placed in machine at each replacement it is not possible to secure tonnage figures for the entire life of any one set of shells.

Lead ores, limerock, slag, iron ore, etc, through No. 1 section from 40 per cent. of the total tonnage through the mill.

Copper ores, the greater percentage of lead matte, lining ores, etc., go through No. 2 section, forming 60 per cent. of the total crushed.

FIG. 10.—ROLL SHELLS, 30 IN. DIAMETER BY 14 IN. FACE, $3\frac{1}{4}$ IN. THICK, NEW; AND SHELLS WORN DOWN TO ABOUT $\frac{1}{2}$ TO $\frac{1}{4}$ IN. THICK. ORIGINAL WEIGHT, 1,200 LB.; SCRAP WEIGHT (ABOUT), 75 LB.

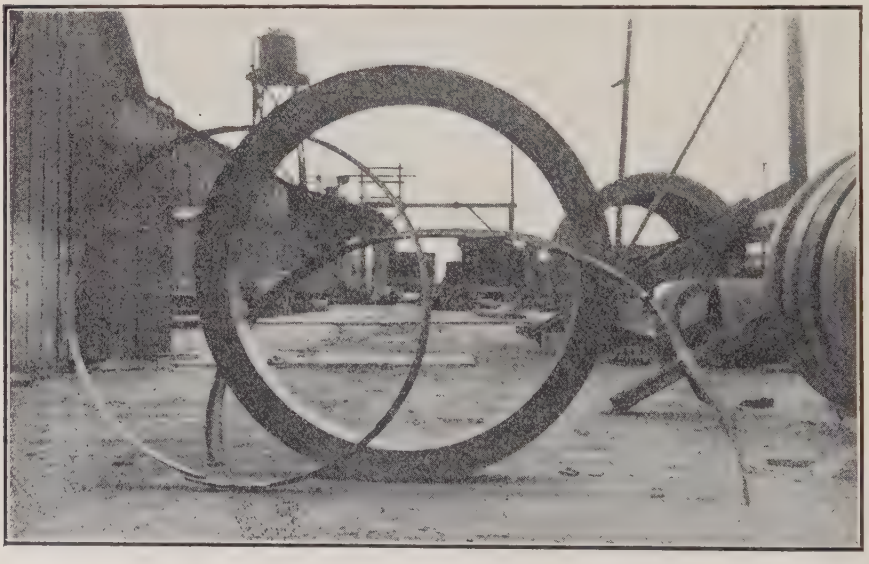


FIG. 11.—CHILEAN MILL TIRES, 55 IN. DIAMETER BY 8 IN. FACE, $4\frac{1}{2}$ IN. THICK, NEW; AND TIRE WORN THROUGH. THE HALF ROUND SHAPED PIECE WAS PULLED OUT BY HAND FROM A CIRCLE. ORIGINAL WEIGHT, 1,450 LB.; SCRAP WEIGHT (ABOUT), 100 LB.

Figs. 10 and 11 are excellent examples of how thin shells should wear in actual operating practice in order to be considered as most efficient and satisfactory. Fig. 10 shows the standard type of straight-bore Cornish roll shell, in this case 30 in. diameter by 14 in. face, worn down to its minimum point. Fig. 11 illustrates rolled tires used on one of the best types of Chilean mills, also worn down so thin that the half-round or crescent-shaped piece shown was pulled out by hand from its circular shape. Both these examples are ocular demonstrations and would prove, more than any amount of elaborate tests or record keeping, that the material of which these particular shells and tires was made, was homogeneous and durable as well as tough, the latter being indicated by its strength and resistance to breaking, even when worn to the thin section shown.

The tabulated list of weights compiled by the writer, for the various sizes of roll shells given below, may be of interest in a general way in connection with this article.

Weights of Rolled-Steel Shells

(*Straight Bore*)

	Diameter in Inches	Face							
		6-in., Pounds	8-in., Pounds	10-in., Pounds	12-in., Pounds	14-in., Pounds	15-in., Pounds	16-in., Pounds	18-in, Pounds
2½ in. thick	12	126	168	210	252	.....	.....	.....	.....
	18	205	270	335	410	.....	.....	.....	.....
	20	230	300	375	460	630	675	.....	.....
	24	285	380	475	570	780	840	.....	.....
	26	310	415	520	730	850	915	975	.....
	27	325	435	540	760	890	1,135	1,300	.....
	30	430	575	715	860	1,005	1,375	1,465	1,650
3 in. thick	36	525	700	870	1,050	1,225	1,690	1,805	2,030
	40	590	785	980	1,180	1,375	2,000	2,030	2,285
4 in. thick	42	805	1,070	1,340	1,610	1,880	2,015	2,145	2,415
	44	850	1,130	1,415	1,700	1,985	2,125	2,270	2,550
	48	930	1,240	1,550	1,860	2,190	2,345	2,500	2,790
	54	1,060	1,415	1,770	2,120	2,475	2,655	2,830	3,180
	56	1,100	1,470	1,840	2,200	2,570	2,755	2,940	3,300

My thanks are due to the operating heads of the mining companies cited for the supplying of data of performance from their records, also to the officials of the steel works for moral assistance in the compilation of this article.

Finishing Temperatures and Properties of Rails*

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(Pittsburgh Meeting, October, 1914)

Summary of Researches and Conclusions

The main objects of this investigation were to determine, from measurements taken at representative rail mills, the present American practice regarding the temperatures at which rails are rolled; to demonstrate the ease and accuracy with which such temperatures may be measured, without interfering in any way with the manufacture of the product; to find out what the "shrinkage clause" in rail specifications really means; and, finally, to determine some of the physical properties of rail steels, particularly those of interest in manufacture, some of which seem to be insufficiently known as yet.

1. *Ingot Temperatures.*—Measurements made in four mills show that ingots for rails are rolled at temperatures ranging from 1,075° to 1,150° C. (1,965° to 2,100° F.), and that, in a series of 20 to 40 ingots, the variation in temperature from one ingot to another is only 10° to 20° C.

2. *Finishing Temperatures.*—Rails are finished at temperatures ranging broadly from 880° to 1,050° C. (1,615 to 1,922° F.), but usually ranging within 50° C. of 935° C., and occasionally above 1,100° C.

3. *Finishing Temperatures of Different Rail Sections.*—With uniform mill practice, the rails of 100-lb. section will be finished at a temperature about 10° to 20° C. hotter than rails of 90-lb. section, and about 50° C. hotter than those of 75-lb. section.

4. *Uniformity of Finishing Temperatures.*—It is possible to roll rails so that all rails from the same original position in the ingot—as *A* or *B*, etc.—will be finished within ± 10° C. of the same temperature.

5. *Temperature Measurements.*—The finishing temperatures of *each* rail may be measured at the hot saws, by means of suitable optical or radiation pyrometers, without interfering in any way with the manufacturing operations.

* NOTE BY THE EDITOR.—This *résumé* of *Technologic Paper No.* 38, which is published in full by the U. S. Bureau of Standards, is brought before the membership of the Institute with the object of affording an opportunity for the open discussion of results of an investigation by the authors named above, of same phases of the manufacture of steel rails which are of great importance to manufacturer and consumer alike.

6. *"Shrinkage Clause" in Rail Specifications.*—A comparison of the shrinkage clause in American rail specifications (for example, those of the A. S. T. M.) with the expansion of rail steel shows that this clause permits rails to be finished at temperatures as high as 1,120° C. (2,045° F.), which is 450° C. (840° F.) above the critical range of rail steel, and which is also above the temperature at which many ingots for rails are actually rolled in practice. Such a shrinkage clause, therefore, does not serve the avowed purpose of limiting the finishing temperature to a value slightly above the critical range.

7. *Critical Ranges.*—For the 10 samples of open-hearth and Bessemer steels examined, the maximum point of the critical range on heating was determined to be within 7° C. of 732° C. (1,350° F.); the critical range on cooling was determined to be between the limits of 680° and 650° C. (1,256° and 1,202° F.).

8. *Melting Range.*—The melting, or freezing, range for rail steels extends from about 1,470° C. (2,680° F.) to nearly the melting point of iron, located at 1,530° C. (2,786° F.).

9. *Expansion.*—The expansion for open-hearth and Bessemer steels is not the same. Above 800° C. (1,470° F.) the expansion of both increases linearly with temperature. The linear coefficient per degree centigrade has the following mean value between 0° and 1,000° C.:

For Bessemer steel (carbon 0.40 to 0.50 per cent.), $\alpha = 0.0000146$.

For open-hearth steel (carbon 0.65 to 0.70 per cent.), $\alpha = 0.0000156$.

Shrinkage measurements on open-hearth rails (C = 0.71 per cent.) gave $\alpha = 0.0000161$.

10. *Rate of Cooling.*—A rail of 100-lb. section, cooling freely in air from a uniform temperature of 1,070° C. (1,960° F.), reaches its recalescence point (670° C. = 1,232° F.) in about 8 min. 30 sec. The maximum difference in temperature between center and outside of head during this cooling is about 85° C. at 1,000° C. (1,832° F.), becoming 0° again at 670° C.

11. *Other Observations of Properties.*—Chemical, microscopic, and mechanical tests were made on a number of samples of rails whose temperatures of rolling had been observed, but there appears to be an insufficient degree of correlation to warrant associating very specifically any of the characteristics defined by these three methods of examination, either with the temperatures of rolling here observed, or with each other.

Observations on the Temperatures of Rolling

The results of the observations on the temperatures of rolling at the four different mills are summarized in Table I. The only considerable element of non-uniformity of the measurements at the several mills lies

in the fact that the distance and time from the finishing pass to the hot saws, at which latter point the observations were made, is not the same. The time for mill *D* is about one-half of that for mills *A*, *B*, and *C*, while the ratio of distances is roughly *A*, 4; *B*, 5; *C*, 3; and *D*, 1.

Importance and Possibility of Temperature Control of Rail Rolling

The undesirable effects of rolling rails too cold and too hot were very early recognized, and have been stated many times by competent authorities, although it should be pointed out that there is not complete agreement, or conclusive evidence, regarding the effects of finishing rails too hot. The argument is that rolling below the critical range distorts and weakens the crystalline structure, while rolling and finishing at too high temperature above the critical range produces a coarse-grained steel, which is weaker than the finer-grained structure obtained by doing work continuously down to the critical range. Indeed, the researches of Wickhorst[1] showed that, with increase of finishing temperature, there was a slight decrease in ductility and a notable increase in the size of the grain.

The introduction of temperature control in rail mills presents no unusual difficulties from the pyrometric point of view. What is desired is a measure of the actual temperature of each of the rail bars as they emerge from the finishing pass, or, better, of the blooms as they enter the rail mill. The present shrinkage clause in rail specifications has no significance whatever, since it is placed so high as practically not to limit the upper rolling temperature at all. It is furthermore an unsatisfactory and inaccurate gauge, and does not permit the exercise of control of the operation, because it can only be used after the rails are completed and cold. This last objection also applies to the determination of the temperature of rolling by means of microscopic examination.

There are many types of optical and radiation pyrometers available to solve the problem of direct temperature control. These pyrometers have given excellent service in other technical fields, and with very small cost of installation and maintenance. In the researches made by the authors of this paper all observations were made without interference with the regular operation of the mills, and no special arrangements or installations were found necessary for manipulating the pyrometric apparatus.

Observations on the Microstructure of the Rails

In nearly all the cases the size of the pearlite grains shows that the rolling was discontinued at a temperature well above the critical range,

[1] M. H. Wickhorst, Influence of Rolling Temperatures on the Properties of Bessemer Rails, *Report No.* 21, *to Rail Committee, American Railway Engineering Association* (November, 1911).

TABLE I.—*Summary of Ingot-Rolling and Rail-Finishing Temperatures in Centigrade Degrees*

	Mill A[a]			Mill B[a]	Mill C[t]	Mill D[t]	
Average temperature rolling ingots	1140[a]	1082[b]		1102[c]	1087[d]	118[e]	
Average time in blooming mill, seconds	120	121		91	71	37	
Average temperature of finishing rail	924[f]	976[f]	988[k]	939[n]	911	1047[n]	992[n]
	918[g]	964[g]	962[m]	928[p]	928[f]		
					901		
					920[g]		
	906[i]	942[i]			883		
					904[i]		
Number of rails observed	120	70	80	312	171	186	
Weight of rails, lb.-section	75	90	100	72	90	100	
Type of rail	Bess.	O.H.	O.H.	O.H.	O.H.	Bess.	O.H.
Location of point of observation	Hot saw			Hot saw	Cooling Bed No.1	Hot saw	
Distance from last pass to point of observation in arbitrary units	4			5	3	1	
	Blooms not reheated			Blooms not reheated	Blooms reheated	Blooms reheated	

[a] Observations taken on 20 ingots; maximum variation from average ± 10°
[b] Observations taken on 19 ingots; maximum variation from average ± 16°
[c] Observations taken on 30 ingots; maximum variation from average ± 12°
[d] Observations taken on 29 ingots; maximum variation from average ± 17°
[e] Observations taken on 43 ingots; maximum variation from average ± 15°
[f] Rails located in ingot at *A* and D
[g] Rails located in ingot at *B* and *E*
[i] Rails located in ingot at *C* and *F*
[k] Rails located in ingot at *A* and *C*
[m] Rails located in ingot at *B* and *D*
[n] Rails located in ingot at mean of *ABCD*
[p] Rails located in ingot at mean of *EFGH*
[s] Blooms not reheated before rolling into rails
[t] Blooms reheated before rolling into rails.

but the results here obtained do not bear out the contention that the finishing temperature is equivalent, so far as grain size is concerned, to the highest temperature of objects heated but not rolled or forged. On the contrary, the grain size is smaller than would have been produced by heating the same steel to the temperature at which rolling was discontinued, and then allowing it to cool without mechanical work. It would appear probable then that the restoration of the crystalline arrangement, which is so badly shattered by the squeezing action of the rolls, is not so rapid as is generally supposed, and, although the restoring process begins immediately after the rolling ceases, it is not complete until the steel has cooled somewhat, which, in fact, produces an "effective finishing temperature" somewhat below the observed one.

From the data in the possession of these authors it would appear that the grain size of the steel is determined as much by the total amount of reduction in the rolls as it is by the finishing temperature, or any other factor.

For the determination of any numerical relation between the size of grain and the mechanical properties of the steel a study of the same steel under varying conditions is essential.

The effect of manganese in retarding the condition of equilibrium during cooling has an important effect on the structure and properties of steel.

Discussion

Albert Sauveur, Cambridge, Mass.—The Bureau of Standards has once more rendered a public service. Manufacturers and consumers of steel rails alike should welcome the important results now laid before them by Dr. Burgess and his co-workers. Personally I welcome these results all the more that in the past, as some of the members may remember, I have never failed, when occasion arose, to call the attention of rail manufacturers to the importance of controlling the finishing temperature of their rails. The pioneer metallographic work conducted at the South Works of the Illinois Steel Co. in the early '90s clearly brought out the close relations existing between the manufacture, properties, and finishing temperature of steel rails and I think I may say without fear of being contradicted that the results of this early investigation had considerable influence in drawing the attention of manufacturers to the importance of finishing temperature and of heat treatment in general. The results now before us should be, it seems to me, of great value to rail manufacturers in their efforts to produce a better and, above all, a safer rail. It seems to be demonstrated in this paper that there is no difficulty in installing in rail mills pyrometric outfits, making it possible to record, without interfering in the mill operations, the temperature of the rails as they leave the finishing rolls, or of the ingots as they are withdrawn

from the soaking pits, or at any intermediate point in the process of manufacture. The inference necessarily follows that manufacturers should see to it that such pyrometric installations are made, in order that they may actually know, instead of guessing at, the temperature of their rails; and, for obvious reasons, rail consumers should insist upon such installations.

In the rolling of heavy sections especially should care be taken to lower their finishing temperature, seeing that the present tendency is to finish them at higher temperature, whereas, owing to the slower cooling of their central portions, which makes for coarseness of grain, they should, if anything, be finished at a lower temperature than lighter rails.

It also seems as if the authors had demonstrated the utter uselessness of the shrinkage clause as a means of regulating finishing temperature. When we are shown that the shrinkage clause allows rails to be finished at a temperature as high as 1,120° C., or some 450° above their critical range, it is high time to discard it, not only as useless but actually as dangerous.

R. Trimble, Pittsburgh, Pa.—One of the most perplexing things we have had to do with in connection with the manufacture of rails is to secure their rolling at a proper temperature, and, as stated in the paper, the railroad engineers have endeavored to have rails rolled at a lower temperature, thinking that such rails would give better service, while on the other hand the mills are favorable to a high temperature, on account of the reduction in wear and tear on the mills due to rolling the rails at a higher temperature.

Although we have in our specifications a shrinkage clause, which we think should be readily met by the mills, yet one mill insists that it is impossible for it to reach our shrinkage clause, and we therefore have to make an allowance to this particular mill, which is objectionable.

William R. Webster, Philadelphia, Pa.—The authors have given us some very valuable information on temperatures at which rails are rolled at our mills in every-day practice in the execution of orders under requirements of ordinary specifications. The results of their work will, as a matter of course, lead to a very thorough and complete investigation of the whole subject. The Bureau of Standards is well fitted to undertake this work and the members of this society should co-operate with them in every way possible.

The present paper forms a very good reopening of our former discussions on the physics of steel and it is hoped that the authors will favor us with further data, from time to time, in order that definite conclusions may be arrived at on the several points they have raised, and on others they have not referred to, but which are well recognized as having an

important bearing on the subject and will undoubtedly be brought out in the discussion.

The practical mill men should be consulted and their views obtained in this investigation, as it is more complicated than at first appears. For instance, the suggested plan for controlling and securing the best finishing temperatures in rolling cannot be carried out in practice, owing to the section of our T rails. In rolling rails direct from the ingot, the ingot is necessarily hotter than is desirable, in order to carry the heat through to the final passes and avoid finishing the flanges too cold. In rolling rails from reheated blooms, it is impossible to secure the beneficial effects of such reheating, as the blooms are heated higher than is desirable, in order to carry the heat through to the final passes. This investigation will develop that it is not merely that the manufacturer "prefers to roll his rails hot, as less power is then required to operate the mill and there is less wear and tear of the rolls," but that with the present sections he is not only compelled to roll hot, but he cannot finish the heads at temperatures low enough to produce the best structure in the steel.

It is therefore suggested that the authors repeat their observations on similar lots of rails rolled at all four mills referred to, under their own supervision, in order to see at just how low temperatures they can start with the ingot or bloom and how cold such rails can be rolled to maintain section and comply with drop-test requirements. At mills C and D, where four saws are used, the shrinkage of each rail should be measured in order to compare the variations with the observed temperatures of each rail taken by the pyrometer. It would also be well to note the amount of camber put in these rails in order to see what effect the cambering has upon the shrinkage and to determine definitely what the coefficient should be for both open-hearth and Bessemer steels.

The authors assume that their coefficient of contraction is correct and, based on it, state that the present shrinkage clause allows "rails to be finished at 1,120° C., or 450° C. above the critical range of rail steel." They also state, in connection with their own work: "From the thermal data, it follows that all these rails could have had work done on them to advantage during the rolling process down to 700° C. (1,292° F.), whereas for none of them was the actual finishing temperature lower than 880° C. (1,615° F.)." This finishing temperature of 880° C. at which some of the rails were finished corresponds to a shrinkage of about $5\frac{7}{16}$ in. in 33 ft., when figured by the same coefficient of contraction. Did the authors measure the contraction of any 100-lb. rail finished at this temperature, and if so, what was the actual shrinkage? The finishing temperature of 700° C. would only give a contraction of about $4\frac{3}{8}$ in. in 33 ft. when estimated by the same coefficient. Do the authors consider that any rails of the present sections can be rolled at low enough temperatures to give only about $4\frac{3}{8}$ in. shrinkage in 33 ft.? We have here the desired finishing

temperature on the one hand and on the other the rolling difficulties owing to the present sections of rail. How are these to be brought into accord without adding metal to the web and flanges to permit the rails being finished at the proper temperatures? How much metal should be added to the web and flanges and how close would such modified sections approach to the English bull-headed rail?

The writer's views on this phase of the question were given in 1900 in an informal discussion at the annual convention of the American Society of Civil Engineers, in London, on Recent Practice in Rails. In March, 1901, he asked the Society to appoint a committee on rail sections to consider modifying the Society's sections by putting more metal in the flanges to carry the heat and allow lower rolling temperatures, and claimed that the rolling temperatures were of as much importance as the chemical composition. His views were also given in a supplemental report to the final report of that committee in 1910, and again in his recent Discussion on Rails before the American Railway Engineering Association in March, 1913.

Too much stress cannot be placed on the influence of the finishing temperature on the quality of the steel; but formerly its effects were not well known or appreciated. The writer's attention was first called to the great influence of the initial rolling temperatures, amount of reduction in each pass, and finishing temperatures, in his investigations on The Relations between the Chemical Constitution and Physical Properties of Steel in 1891 to 1894. It was then found that the control of the rolling temperatures was the most important factor. Since that time the writer has been greatly interested in the effect of the heat treatment on all grades of steel in forging and rolling and, as far as he knows, was the first to propose the shrinkage clause in rail specifications. He has always appreciated its limitations and the force of the objections raised from time to time, but does not consider that any practical substitute has yet been offered. It was the first effort made by the purchasers of rails to systematically control and check the finishing temperatures of their rails. The rail-mill practice prior to that time has often been referred to as "hardening the steel by the addition of carbon and squirting it through the rolls in order to get increased output." It is hoped that the authors will give us further data showing how accurate a measurer of temperature the shrinkage really is, and what reductions in the present specified shrinkages can consistently be made for rails of present section.

Assuming the pyrometer can be used as suggested, in every-day practice, no explanation has yet been offered as to how the individual rails on which observations have been taken can be identified in the finishing department; but the shrinkage of the rail leaves a permanent record. All the inspector has to do at mills where four saws are used is to measure the distance between saws and the lengths of the cold rails. This check

on the finishing temperature by the shrinkage is therefore automatic in every respect.

There are two problems that come into this matter: First, public safety; second, wear of our rails.

If you roll your rails of present sections too cold, endeavoring thus to secure better wear, the flanges will be finished too cold, causing the steel to tear and thus forming small fractures, many of which are covered up in rolling and are not observable on inspection. These incipient flaws in the flanges of our high-carbon rails are much more dangerous than is generally appreciated. We should therefore go very slowly in this matter before coming to any final conclusions on cold rolling of rails until sections are decidedly changed.

P. H. Dudley, New York, N. Y.—

1. *Introduction*

The periodic recurrent discussion of one or more phases of rail manufacture emphasizes:

1. That the service of rails has increased in the track in a faster ratio than generally realized.

2. The complicated problems of rail manufacture for present service in the United States are but partly encompassed by metallurgic knowledge and practice, but include those of mechanical and civil engineering, operations, climatic conditions, and a combined mill and track experience.

Page 4[2] of the Introduction states: "It is common knowledge that there are an alarming number of failures of rails in service on the railroads of the country, and, unfortunately, these failures are not on the decrease. Thus the reports of the rail committee of the American Railway Association show 36,641 rail failures, in 12,688,714 tons laid, for the year ending October 31, 1911, and 61,047 failures for 13,736,956 tons [October 31] in 1912."

The increase of failures from 36,641 in 1911, to 61,047 for an addition of only 1,048,242 more tons reported (October 31) in 1912, revealed the fact that there was some decided change in the conditions of operation which contributed to lessen the duration strain factor of the metal in the rails to a greater degree for the low temperatures in 1912, than for a milder period in 1911.

The winter of 1911 was mild for most of the northern railways, but 1912 was extremely cold, with the greatest number of broken rails in their history; 1913 was mild, with 60 per cent. reduction over 1912 of broken rails, while 1914 was again cold in the eastern portions of the United States and Canada.

[2] *Technologic Paper No.* 38, *U. S. Bureau of Standards* (1914).

November, 1911, was cool, with a range of 50° to 120° deficiency temperatures for the month at different U. S. Weather Bureau stations. December had an opposite oscillation of warm waves of 100° to 200° above normal. Rails which had contracted in the splice bars during November, were again expanded in December. The sudden excessive cold wave in the West of the last one or two days of December, and the first day of January in the East, contracted the metal in the rails and set up tensile stresses before the rails could render in the splice bars. This, combined with the stresses of the metal of the rails as girders under the moving wheels of the locomotives and cars, was the important factor in 1912 of the greatest number of breakages of Bessemer rails in the history of the railways.

The ordinary duration strain factor of the metal of more Bessemer rails than usual was reduced from the effects of the cold by:

1. The additional tensile stresses due to being held by the splice bars from contraction.

2. The increased tendency of the metal of Bessemer rails of 0.10 phosphorus and only 0.50 carbon to fracture under rapid distribution of the alternate unit fiber strains and balanced stresses of the moving wheel loads.

I sent out September, 1913, 450 copies of the tabulations of the accumulated excess or deficiency temperatures for the United States Weather Bureau stations on the New York Central Lines for the years 1911, 1912, and 1913 to Aug. 31, to the respective officials in charge of maintenance of way and to those of equipment.

I made the forecast September 1, that the probabilities were that the winter of 1914 would not be as mild as the winter of 1913—the broken rails not being one-half as many as in 1912 for the same locality and service.

I also sent instructions to have the maintenance of way to its highest standard in 60 days, and the equipment, as to the rotundity of the wheels, in the next 90 days, and to be maintained through the winter.

The forecast proved true, though December and the fore part of January were mild; the cold wave was severe in the latter part of January, February, and March, but from the care and attention of the officials of the respective departments, the percentages of broken rails and shelled wheels were less than in normal years.

The excess or deficiency temperatures per month for 1914, to August 31, have been added to the three preceding years, with forecast and instructions issued for the winter of 1915, and copies sent to the officials in charge of the departments of maintenance of way, of equipment, and operating. These three contribute by their care to the safety limits of the duration strain factor in the metal of the rails.

The large railway systems keep their own statistics up to date, there-

fore know from the comparative figures that the rail failures have decreased since the extreme cold winter of 1912, and are proportionally less than ever before for a similar service. I speak particularly for the New York Central Lines, for in some territory they often have cold waves of some days' duration of 30° and 40° F. below zero.

The introduction of basic open-hearth rails the past few years in which the phosphorus content is practically one-fourth of the former Bessemer rails, furnishes the metal of the sections as girders a duration strain factor of greater combined capacity and rapidity of distribution of the alternate unit fiber strains, balanced by equal stresses, than was possible to secure in many of the 0.10 phosphorus and only 0.50 carbon Bessemer rails.

The basic open-hearth rails for the New York Central Lines have all been made since 1910 under the 1909 specifications of their elongation and exhausted ductility tests which were developed from the elongation tests under the drop used for Bessemer rails from 1890 to 1910. There were in the tracks of the N.Y.C. & H.R.R.R. Jan. 1, 1914:

Tons	Basic Open-Hearth Rails		Length, Miles	Broken per 100 Miles
65,297	5⅛ in.	80 lb.	519	1.93
83,268	6 in.	100 lb.	530	0.57
67,460	6 in.	105 lb.	408	0.00

There were 13 broken basic open-hearth rails from Jan. 1 to Aug. 1, 1914, and 279 Bessemer, or 1 open-hearth to 22 of Bessemer, exclusive of interior transverse fissures. February and March were extremely cold, with many continuous cold waves of 35° F. below zero and an occasional wave of 40°.

2. *Importance of Limiting Temperatures of Rolling Rails*

This is a comprehensive subject and will require many papers and discussions to state its many salient features, for application to rail manufacture for the girder and resistance to wear.

I state here that the effects of the rolling temperatures upon the duration strain factor of the metal of the basic open-hearth rails as girders, for service either in warm or cold climates, is one of the keynotes of this discussion. Investigations have shown that the transformations of all the different metallographic forms the metal should pass through from the higher temperatures of the hot set ingot to the lower of the metal of the rail in the finishing pass, did not completely occur in the metal of the section, particularly of that in the central portions of the head. It is left in a heterogeneous condition in a few rails with initial strains, and as the interior metal cannot elongate as readily as its exterior envelope of the

head, is sometimes checked by the gags of the presses to straighten the rails. These develop in service from the negative strains in the wheel spacing of the passing equipment.

3. *Rolling Rails*

Rails are rolled:

1. Direct from the equalized, original heat of the ingot to the finished rail.

2. The ingots are bloomed from their equalized, original heat, then the blooms are reheated to pass through the roughing and finishing roll trains.

The investigations show that as a rule the transformations seem more complete in a greater majority of rails from the reheated blooms, than in direct rolling. This is a difficult investigation to make direct upon the output of the rails owing to the small fraction of 1 per cent. affected. The fact is confirmed by only an occasional failed rail from the reheated blooms due to the incomplete transformations.

There have been two distinct schools of theory and practice for the manufacture of rails:

The first considers that the chemical composition should be prescribed and limited by definite proportions to make sound hot ingots, combined with rolling temperatures to secure the physical properties intended in desired sections of girders.

The second considers the chemical composition secondary, but that the rails must be rolled cold to insure good wear, as the only information to be given the manufacturer to secure a suitable product.

The first school has service tests of rails in the tracks for over two decades which proved safe and quite free from breakages in low temperatures, with fair resistance to wear. This has been of great service in the adaptation of basic open-hearth steel to replace Bessemer.

The second school had the mills remodeled for cold rolling of Bessemer rails (some were built for the purpose); and prescribed shrinkage limits for the hot saws, which have not been modified for basic open-hearth rails.

The holding the bars 40 sec. before the last pass, was abandoned many years ago, when the epidemics of broken rails in the low temperatures of 1904, 1905, and 1906 occurred. The mills which were built for direct cold rolling for wear, without the least intention of so doing, injured many rails of their output of basic open-hearth rails as girders.

The second school is no longer advocating the extreme cold rolling, for the service tests have not sustained its contention.

The metal of a rail section, as an element in the "Means of Transport" to carry and guide the passing equipment, performs two diverse functions:

1. That of a girder, to distribute in fractions of a second large alternate

unit fiber strains, and which, for safety, governs the quality of metal which can be used.

2. That of resistance to wear and abrasion from the passing wheels of the equipment.

4. *Interior Transverse Fissures Due to Incomplete Transformations of Metal from the Higher Temperatures of the Ingot Metal to That of the Finished Rail*

It does not seem to matter what the distribution of metal is in the section, whether of thin or heavy bases, for each has developed interior transverse fissures in the head after more or less service. A direct connection seems to be established with cold direct rolling temperatures in retarding the transformations, which must receive careful research before it can be determined what is the proper range of finishing temperatures for the combined functions of a given section and its service. It may fail from an interior transverse fissure either in a warm or cold climate.

The paper is a valuable contribution of data, and I concur with the authors that more experimental work should be done, and the rails subjected to service tests to establish the important facts as to the finishing temperatures and properties of rails.

George B. Waterhouse, Buffalo, N. Y.—The paper of Dr. Burgess and his associates is one of great interest alike to the theorist, mainly concerned with pyrometry and its application, and to the practical man, whose chief interest is improvement and uniformity in his product. The comparative ease with which the temperature measurements were made is specially noteworthy, and shows that the newer types of optical pyrometers must be workmanlike instruments that can be depended upon even when subjected to the hard and exacting conditions found in steel works and rolling mills.

While the paper as a whole is valuable, several of the methods used and results obtained are open to criticism, and it is hoped that the following remarks will be accepted as a sincere effort to help in the study of this important subject of rail shrinkage and finishing temperatures.

In the first place, it seems to me that the relation between finishing temperature and shrinkage has been arrived at in a needlessly indirect way. The temperatures were taken directly in the mills, and it would have been very easy to obtain the exact shrinkage on the same rails that were observed. Instead of this, pieces were cut from rail flanges, welded together either by electricity or oxy-acetylene, and the coefficient of expansion obtained carefully and laboriously in the physical laboratory. Having obtained this coefficient, the finishing temperature corresponding to a given shrinkage was worked out. The results so obtained are questionable on account of the influence of the welded joints. It would

not have been hard to get pieces from the mills of the required length, about 20 in., and it is to be regretted that this was not done.

According to these results there is quite an appreciable difference in the expansion of open-hearth and Bessemer rail steel, which would cause Bessemer rails to be finished about 100° F. hotter than open-hearth in order to have the same shrinkage. Regarding this they write: "This great difference in the shrinkage of the two classes of rails does not appear to have been considered in rail specifications, although this difference must have been detected in mill practice." As a matter of fact there is very little difference in the shrinkage of the two classes of steel. At our mill, Bessemer rails 100 lb. to the yard, 33 ft. long, require 6½ in. at the hot saw, while open-hearth rails of the same section, rolled and finished at practically the same temperature, require as a rule from 6½ to 6⅝ in.

In regard to the temperature measurements one thing should be carefully noted, and that is that they were chiefly made on the head of the rail, which is the hottest part of the rail section. The ends of the flanges are finished much colder even with the heavy-base rails, and this difference is particularly marked with thin-flange rails such as the New York Central uses. It is obvious that the finishing temperature of every part of the rail section must be well above the critical range so as to avoid brittleness, and it is certain that this will cause the rail heads to be finished a good deal hotter than is regarded as sufficient in the paper—that is, about 1,300° to 1,350° F.

The section on microstructure is very interesting, although it is stated that no definite conclusions can be drawn as to the relation between grain size and finishing temperatures. One conclusion is arrived at, however, that appears to be definitely proved from the known evidence on the subject, and which is very valuable. The paragraph may be quoted. "It would appear probable, then, that the restoration of the crystalline arrangement which is so badly shattered by the squeezing action of the rolls is not so rapid as is generally supposed, and though the restoring process begins immediately after the rolling ceases it is not complete until at a lower temperature, which in fact is the temperature which determines the grain size of the finished product."

In other words, the effective finishing temperature as it concerns the properties of the finished rail is below the temperature as observed by the pyrometer by an amount that is not thoroughly determined, but which must be allowed for. The temperature, therefore, on leaving the rolls must be sufficiently above the critical range to allow the restoring process to get well on its way in all parts of the rail section.

The paper also lays emphasis on the influence of the amount of reduction on the grain size, and the greater the reduction the smaller the grain size. This does not refer to the number of passes but to the actual re-

duction from the cross-section of the ingot to that of the finished rail. In this connection it does not seem as if enough importance has been accorded to one factor governing the grain size: namely, the rate of cooling. The larger the finished rail the larger the section of the various passes from the bloom to the rail, and therefore the slower the rate of cooling on the mill tables and the hot bed. To follow this further, the slower the cooling the higher the finishing temperature, and the greater the time interval between the effective finishing temperature and the critical range. This will inevitably bring about a larger grain size in the finished rail.

Finally, in regard to the difference in temperature between the inside and outside of the head of a rail, it is probable that the difference in practice is greater than that found by the writers in the laboratory. It must be remembered that the rail ingots are charged into the soaking pits while the interior is very little if at all below the freezing point, while the outside is usually below a good rolling temperature. Even after the customary time in the pits the interior is undoubtedly hotter than the outside, and this difference is increased during rolling because of the cooling on the tables and the water used on the rolls. This difference in temperature is important because of the shrinkage stresses that are liable to be set up, and which must be carefully considered.

M. H. Wickhorst, Chicago, Ill.—I have made a few temperature measurements, and, although Dr. Burgess has described me as an optimist on the subject, I have not felt all the confidence in the correctness of the measurements that I would have liked, and the Bureau can, I think, do considerable toward advancing the art of measuring rolling temperatures to a point where we can accept such measurements with confidence.

I wish now to make a few remarks on the subject of the relation of the rolling temperature to the general question of rail failures. Some years ago it was customary to ascribe rail failures to too high finishing temperatures of the rail, necessitated by the thin flange, but it now seems that not many failures can be traced to this cause. Very briefly I may say that a large portion of rail failures have their origin in excessive segregation, mostly of carbon and phosphorus, in the interior and upper part of the ingot. Another large portion of rail failures have their origin in seams in the bottom of the base. Excessive segregation is to be avoided by using quiet-setting steel, well deoxidized with silicon, titanium, or aluminum; in other words, ingots with "horny" tops should not be used. Of course, such quiet-setting steel pipes deeply and successful use of such steel will then, perhaps, require a large discard, or perhaps the development of a commercially available sink-head process of casting the ingot. The surface seams, of which those at the bottom of the base

are especially harmful, seem to originate in the tearing of the sides of the ingot in the early stages of blooming, and probably a full study of this part of the subject can be expected to result in the elimination of a great many of the seams.

Having obtained an ingot of fairly even composition, and having given close attention to the rolling of the bloom, it would then seem to be necessary to give attention to the matter of rolling temperature in order to still further improve the properties of the rail.

Dr. Burgess found a considerable difference in the hot shrinkage between Bessemer and open-hearth rails, amounting, I believe, to something like $\frac{1}{2}$ in. difference in a standard rail length of 33 ft. My own work has indicated that the shrinkage of the hot rail increases as the carbon in the steel increases and the difference between steel of average Bessemer carbon of 0.50 per cent. and steel of average open-hearth carbon of 0.69 per cent. would amount to about $\frac{1}{4}$ in. (See *Bulletin of the American Railway Engineering Association*, vol. xvi, No. 170, pp. 163 and 164, October, 1914. It is also expected that this article will appear in the *Proceedings of the American Railway Engineering Association* for 1915.)

Dr. Burgess calls attention to the point that the shrinkage of the rail is not a correct measure of finishing temperature, and this I think we can concur in. At the finishing temperature the rail shrinks at the rate of about $\frac{1}{100}$ in. per second for a standard length of 33 ft. of cold rail, and of course the amount the rail shrinks after leaving the saws will vary according to the length of time elapsed between the finishing rolls and the saws.

As regards the question of grain size, we may ask several questions: To what extent is the grain size a function of finishing temperature; to what extent is it a function of reduction in rolling; to what extent is it a function of the length of time the ingot has been in the pit or the bloom has been in the reheating furnace; or again, to what extent is the wear of a rail a function of the grain size, or of the finishing temperature? These are some of the questions that need attention and can be finally worked out only after a considerable amount of experimental work.

James Aston, Cincinnati, Ohio.—I have been a consistent advocate of the doctrine of the correlation of grain size and physical quality; that, other things being equal, the strength of a steel is inversely proportional to the size of grain. While it is unwise to attempt to draw conclusions from incomplete data, and in the present instance it may be somewhat unfair, I was interested in reconciling the results of the paper under discussion with the above doctrine, and was surprised and disappointed to find that the expected confirmation is lacking.

Upon arranging the data of the several tables of the paper in the order of increasing grain count, together with in each instance the carbon con-

tent, physical quality, weight of section, finishing temperature, mill practice, etc., certain conclusions seem evident. That the rails representing the materials and practice of the several mills appear to be rather promiscuously distributed, and there seems to be no regular order as to weight of section with respect to grain and temperature. And more conspicuously, that there is no concordant relation of grain size as given by the count, and the finishing temperature of rolling. Again, making due allowances for the effect of carbon, the strength does not follow the order of increasing fineness of grain; and this holds even more noticeably for the relative ductilities. Thus, of course, it is equally true that there is no correlation of physical quality and finishing temperature.

On the face of such returns we can hardly expect the mills to rush wildly into the adoption of lower temperatures of rolling; at least until more consistent and conclusive data confirming the grain-size doctrine are available.

With respect to the statement of the paper that the rolling operation produces "an 'effective finishing temperature' somewhat below the observed one" and "it would appear that the grain size of the steel is determined as much by the total amount of reduction in the rolls as it is by the finishing temperature, or any other factor," a logical answer seems to be, "Why not?" We grant that mechanical working is a factor in reduction of grain size. Consequently any reduction in the later stages of rolling, and especially that in the last or finishing pass, will break the grain down to a size less than that normal to the finishing temperature. This grain will tend to grow to normal during cooling, but will require appreciable time for its consummation; and during such interval the temperature will be falling below that of the finishing pass. So that equilibrium will be reached with a grain growth arrested not at the finishing temperature, but at one somewhat below. The spread will be dependent primarily upon the amount of the final reductions, but to a degree also upon the speed of cooling, which in turn is influenced by weight of section, and other factors.

In view of the above, the suggestion of the authors that there may be some advantage in measuring the temperature of the blooms as they enter the rail mill, with the assumption of a constant speed of rolling, would seem open to question. For would we not have other disturbing factors varying with the amount of final reduction, weight of rail section, influence of chemical constituents, details of mill practice, and the like; all making for variations in grain size from that to be expected from temperature conditions alone?

Speaking of rail breakage, the severe winter of 1911–12 resulted in an epidemic of such failures in the Northern States. I recall an instance of the breakage by a single flat wheel of upward of 100 rails on an 85-mile stretch of as fine a piece of roadbed as there is in the country. Are

we not too ready to jump at conclusions in reaching for a cause; to ascribe failure to excess of phosphorus, to incipient fissuring, to coarseness of texture resulting from high temperature of finishing, or like causes? Was not the failure in all likelihood due to the proper combination of conditions; of high phosphorus, of abnormal grain, of increased brittleness of the steel due to the low temperature, of heavy wheel load, and of excessive pounding of the flat wheel—of the above singly or in combination? And especially, was there not the important factor of unusual rigidity of frozen roadbed, which took away its power of yielding, and threw the entire resistance upon the overburdened rail?

E. F. Kenney, Johnstown, Pa.—I read Dr. Burgess's paper, and, by the way, I was with Dr. Burgess part of the time when he made the measurements used in this paper. The thing that impressed me first, aside from the very valuable information that is included, is that, I think, the author is reasoning a little along theoretical lines without considering some of the practical lines that would appeal to those in the rail business. Two points should be determined before we lay too much stress on this question of finishing temperatures.

In the first place, have we any real evidence that lower finishing temperatures would result in betterment to the rails? We have not seen any as yet. Some years ago we rolled some rails with widely different finishing temperatures; those rails were from the same heat and were put in on Horseshoe Curve on the Pennsylvania Railroad, and very careful records of measurements were kept showing the wear. We were not able to differentiate between the hot- and the cold-finished rails in that experiment; that is, as far as wear was concerned.

As bearing on the effect of finishing temperatures on the strength of the rails we submit some recent experience. We started to roll a rail of very heavy section, weighing 125 lb. per yard, which because of its chemistry partly, and of its section, is very stiff. The drop tests required were difficult to meet because of liability to breakage. With a view to getting greater toughness we had some of the rails rolled at a temperature 100° F. lower than our ordinary finishing temperatures, or about 1,550° F., which is rather low for rails. We did not go any lower than that, because we were afraid of breaking the rolls. These temperatures were measured at the center of the base, and the edges of the base were, of course, considerably colder. Under the drop test there did not seem to be any difference between these rails and those which were finished at the ordinary temperature. We are therefore led to believe that within the limits of temperature within which the steel can be rolled, we will not get very material difference, as far as the drop test is concerned, between hot-finished rails and cold-finished rails. But let us look at the other side of this matter.

Even if we were assured that we might get some betterment from very cold-finished rails, a number of difficulties are in the way which I think would be very hard to overcome. In some of our products we strive for a very smooth finish. This is well known over the country as "Gautier Smooth Finish," and is obtained by holding the bars in the course of the rolling, so that at the finishing pass the temperature is so low that the heavy scale does not form, and the result is a bar with very smooth, black finish. Tests on those bars show that they have a finer grain and a higher elastic limit than ordinary bars, but the ductility has been reduced in the same proportion.

Even if we could roll rails under the same conditions, I think we would lose in ductility quite as much as we would increase in other qualities. This cold rolling can be done without great difficulty on simple sections such as rounds, squares, etc., but when it is attempted on any complex sections we almost invariably get into trouble. The bar is likely to twist, and this would be fatal in the case of rails, as they would not be accepted. Our present T rails are of very complex section as far as rolling is concerned.

Taking into account the doubt as to the practical effect of colder rolling and the difficulties attending it, I doubt very much if we are going to get much betterment in our rails, due to this colder finishing temperature.

A. W. Gibbs, Philadelphia, Pa. (communication to the Secretary*).—The starting point or datum line of Dr. Burgess's paper is the critical temperature in the neighborhood of 700° C., 1,300° F. Running through the paper is the idea that rails should be finished at a temperature slightly above this critical temperature; notably at the bottom of p. 31,[3] where it says:

"From the thermal data, it follows that all these rails could have had work done on them to advantage during the rolling process down to 700° C. (1,292° F.), whereas for none of them was the actual finishing temperature lower than 880° C. (1,615° F.)."

Further on in the paper, when dealing with the question of the shrinkage clause as used in the different specifications, the excess of temperature above this critical point, as determined by the shrinkage clause together with the coefficients of expansion determined in this paper, is taken as a measure of the badness of the rolling. I would refer particularly to paragraph on page 62, reading:

"A comparison of the shrinkage clause in American rail specifications (for example that of the A. S. T. M.) with the expansion of rail steel shows that this clause permits finishing rails at 1,120° C. (2,045° F.) or 450° C. (810° F.) above the critical range of

* Received Oct. 13, 1914.

[3] *Technologic Paper No. 38, U. S. Bureau of Standards* (1914).

rail steel, and above the temperature at which many ingots for rails are actually rolled in practice. Such a shrinkage clause therefore does not serve the avowed purpose of limiting the finishing temperatures to a value slightly above the critical range."

While following this is the statement by the author that this is but a preliminary investigation, nevertheless I think it is unfortunate that the desirability of finishing at a point slightly above the critical range should be stated as though it were proved.

Those who are following the question of rail specifications are continually met with the statement that certain mills have rolled at finishing temperatures much below the customary ones and that the service results have not been satisfactory.

The steel people, if they will, are in the best position to offer definite proofs.

Before departing very far from the present shrinkage allowance in the rail specifications under which we are rolling, it seems necessary to have some direct evidence on the following points:

Is a greater fineness of grain than we now secure accompanied by a greater safety against shock, and against the development of that form of detail fracture concerning which so much guessing has been done of late? If we are now getting the desired fineness of grain in certain parts of the rail, such as the flange and the web, is it desirable to lower the finishing temperature still more, so that this fineness of grain will be found in the head? Will this rail of finer structure be less able to withstand the drop test? If so, are we justified in reducing the severity of the drop test, and what will be our standing in court in the event of a wreck from rail breakage, where it has been shown that the rails have been subjected to a lessened drop?

W. A. Aiken, New York, N. Y. (communication to the Secretary*). —In a more or less detailed connection of some 15 years with the study of rail steel and the inspecting of the finished product, I have always contended, recognizing as I did the necessarily antagonistic interests involved between the producer and consumer, happily very much more marked some years ago than existent to-day, that the most important point to be definitely settled first, was the absolute necessity to secure as perfect material as possible whereon to base conclusions, arrived at from the study of its heat treatment. While it may be broadly admitted that good material may be abused during manufacturing processes and poorer material possibly improved by such processes, it is a self-evident proposition, in my opinion, that with the finished product under discussion in this paper, not to grasp the vital necessity of studying the best material obtainable is more or less a waste of time and energy and in direct opposition to the theory of the conservation of the resources.

* Received Oct. 5, 1914.

The absolute necessity of sufficient discard from the ingot to insure sound material from which to roll the finished product has long been recognized in principle and has grown in practice; but until the two diversant interests involved become less of an *impasse* than in the past, every one is wasting energy to a certain extent in studies, no matter how elaborate, of material which the *ex parte* student must insist is not as good as can be procured. So that, while in no way disparaging the good work done in the past, as shown in the files of our various technical societies and markedly brought out by the present work of the associated physicists of the Bureau of Standards, no one interested in obtaining better rails, and confronted with the yearly reports of failures of this material, must close his eyes to further and continued effort to obtain better steel from which to manufacture rails. Meanwhile, of course, all study on the finished commercial product cannot be considered entirely wasted, but the value of heat treatment to prevent injury to good material strikes me as much more important than its value in improving other material, not of the best, which is offered under present commercial conditions.

That the shrinkage clause is at best a poor makeshift, cannot be ignored. That proper control of heat treatment can be accomplished instrumentally, I think is established by the present paper; though long since abstractly recognized as feasible commercially by all investigators not biased by commercial interests.

Personally, I have had sufficient experience in the use of pyrometers to have long since felt assured that their application could be effectively made for the determination of temperatures under ordinary rail-rolling conditions. And, finally, the use of such instruments of precision will, in my opinion, settle practically once for all the fact that better finished product will be obtained at much lower finishing temperatures than are now generally used by manufacturers.

It must be recognized, however, that manufacturers cannot arbitrarily be expected to void purely commercial interests for technical ones, even though these latter may be reasonably expected, if substituted, to furnish better material. There should be no question between producer and consumer thoroughly to try out commercially any method promising as good results as the practical and positive control of heat treatment of rails.

Dr. G. K. Burgess (written communication*) I wish to express my gratification at the interest expressed in the paper by so many competent authorities. The matter that appeared to be of the greatest general interest, and which it is only too apparent is still an open question, is, What are the most suitable temperatures, *all* things considered, for

* Answer to discussion.

rolling and finishing rails? As pointed out in several communications, the answer to that question is necessarily a complex one but it is believed to be a determinable one for any given rail section. The allied question, What is the best rail section? is intimately related to the former, as stated in several of the communications, but the answer to this appears to be even more complex. To neither of these questions does the paper give a reply, although more than one of the contributors to the discussion appears to have obtained the impression that there is here presented an argument for finishing rails at lower temperatures than is at present practiced. Emphasis has been placed, however, on the conclusion from this investigation that the present "shrinkage clause," which was designed to limit the upper temperature of finishing rails, does not accomplish this purpose. Statistics as to rejection of rails under this clause would be of interest. In this connection, the remarks of Professor Sauveur and of Mr. Trimble are of particular interest.

Mr. Gibbs considers "it is unfortunate that the desirability of finishing at a point slightly above the critical range should be stated as though it were proved." Of the two quotations cited by him in support of this inference, the first refers to what we consider to be a generally accepted fact, that work can be done to advantage on cooling steel to nearly the critical range; the second has to do with the inadequacy of the "shrinkage clause" to fulfill the purpose of its framers, which was to restrict the upper limit of rolling to a temperature slightly above the critical range. Whether or not rails should be rolled colder or hotter than is the present practice it was not intended to advocate in the paper, and, as Mr. Gibbs states, "the steel people, if they will, are in the best position to offer definite proofs" of the qualities of rails rolled hot or cold.

The fundamental questions asked by Mr. Gibbs regarding the relation of fineness and uniformity of grain, finishing temperatures, resistance to shock, and breakage in the track, can only be answered by experiments on an elaborate scale; and there is also implied here the underlying questions of weight and section of rail.

The comprehensive communication from Mr. Webster is most welcome and timely, and we are glad to note and acknowledge his valuable contributions to our knowledge of the properties of rails and in particular his urging repeatedly during the past 25 years the systematic study on an adequate basis of the fundamental problems of the physics of steel.

It is of more than retrospective interest to look back at the Suggested Lines for the Discussion and Investigation of the Physics of Steel inaugurated at the Chicago meeting of this Institute in 1893, and to note how many of the queries then propounded still remain unanswered, and this in spite of the nearly boundless facilities of our many iron and steel manufacturing establishments, and the existence of not a few laboratories. One cannot help but feel that many of these questions—some of

which Mr. Webster repeats to-day—might have been answered ere this without sacrificing the efficiency of our mills and probably with great economic gain in many instances.

That the government standardizing laboratory is interesting itself—within its necessarily limited sphere of activity—in some of the fundamental problems of the properties of railway material should, we believe, serve as an added stimulus, however slight, to others in the common endeavor to solve these oftentimes perplexing, and happily sometimes easy problems, but ones the solutions of which have been deferred.

Mr. Webster as early as 1890 to 1893 reached the conclusion from his investigations on The Relation between the Chemical and Physical Properties of Steels "that the control of the rolling temperatures was the most important factor;" and yet, to-day, it would appear that neither he nor any one else can say with certainty at what temperatures rails should be rolled.

Regarding the difficulties to be encountered, some of which Mr. Webster mentions, in the practical solution of this problem, we believe some of them to be less serious than might at first appear. There certainly will be no difficulty, for instance, in identifying individual rails the temperatures of which have been measured. Mr. Webster mentions the procedure, under the shrinkage clause, for the inspector to follow in a mill with four saws; but how about the mill with one saw? This mill also works continuously, but has no automatic shrinkage control.

Again, the question of rail section vs. temperature distribution, although undoubtedly troublesome for rails of thin flange, may perhaps be answered; and we hope to be able to contribute to its solution.

To Mr. Webster's suggestions for more work along more extended lines, for taking into account all the factors entering into the problem of rail temperatures and for co-operation with the practical mill men, we most heartily assent.

To his questions as to effect of section and the practical possibility of rolling rails colder than is the present practice, we must plead ignorance.

In view of the fact that the shrinkage measurements made in the mill and the expansion measurements made in the laboratory agree very well for the few examples given in the paper for open-hearth rails, we are inclined to the belief that these measurements are trustworthy; but much more data on shrinkage and finishing temperatures should be obtained for different kinds of steel and for rails embracing at least all the usual sections.

The communication by Mr. Waterhouse contains many useful suggestions which must be kept in mind in a comprehensive study of the subject of rolling temperatures vs. properties of rails.

Regarding his criticism of our method of determining expansion in the laboratory rather than of shrinkage in the mill, it may be stated that our

original program as planned, and as carried out in the mills, involved no interference with mill practice. He appears to overlook the fact that shrinkage measurements were made, however, in one of the mills as shown on p. 55 of *Technologic Paper No.* 38. The results in shrinkage agreed with those on expansion. The effect of welding the rails can easily be shown to be *nil* as carried out. Mr. Waterhouse points out the importance of the metallographic examination indicating an effective finishing temperature lower than that given by the pyrometer. We believe this to be a field worthy of further investigation.

Mr. Aiken, in the light of his considerable experience, insists on the desirability of improving rail material, with which every one would probably agree; but we also believe in the careful study of the manufacturing processes and properties of the resulting products as carried out under present commercial and technical conditions with the materials in general use.

Mr. Wickhorst points out in a comprehensive manner what he considers to be the sequence of importance in the causes of rail failure, and emphasizes the uncertainty in practice of the effect of finishing temperatures and the need of more experimentation on this and similar questions, with all of which we agree.

Mr. Kenney's statement that 100° F. either way from the "ordinary" finishing temperature makes no material difference in the properties of rails of heavy section is of importance, as well as his account of rails from "the same heat" but rolled at "widely different" finishing temperatures not showing appreciable differences in the track. The detailed and complete account of this, and similar experiments on a more exactly determined basis, involving a sufficient quantity of material to yield "practical" results, would be of interest.

Until this is done, the question of suitable finishing temperatures will remain unsolved, for the "theoretical" man still asks, Why not roll colder? and the "practical" man admits he has no certain evidence one way or the other.

Mr. Dudley has given us in his elaborate communication what is practically a new paper on the subject of the service of rails as influenced by manufacturing, maintenance, and weather conditions. His explanation of the abnormal number of failures for the year ending Oct. 31, 1912, as due to the cold winter is undoubtedly correct. The experience of the New York Central Lines that the Bessemer rails have an overwhelming proportion of failures as compared with open-hearth—or 22 to 1—appears not to be borne out generally. Mr. Dudley does not give, however, the total number of Bessemer rails, so that a strict comparison cannot be made between the failures of the two types from his data on the New York Central Lines. For the year 1912, for the whole country, the ratio of Bessemer to open-hearth rail failures per million tons laid was 5,160

to 2,379—or roughly 2 to 1. It would appear from Mr. Dudley's figures that the present breakage of open-hearth rails on the New York Central Lines is 91 per year per million tons laid, or only about $\frac{1}{26}$ the average for the whole country in 1912; from which one would infer that the breakage is in general unnecessarily high, and one should perhaps be cautious in attributing such wholesale failures too readily to increased train loads and high speeds. The questions of maintenance of track and quality of the rails and rolling equipment are perhaps of at least equal importance—factors which Mr. Dudley emphasizes.

Regarding the question of best rolling temperatures, Mr. Dudley appears to consider the unsatisfactory experience with the Kennedy-Morrison process of holding the rail bar 40 sec. before the last pass, to have demonstrated the practical unsuitableness of colder rolling. The fallacy involved in that pseudo-cold-rolling process was explained at the time of its introduction by Professor Sauveur. Mr. Dudley's conclusion concerning shrinkage limits prescribed by the school favoring cold rolling loses some of its force in view of the fact that the shrinkage limits have almost without exception permitted rolling at very much higher temperatures than has apparently been generally supposed.

The question of transverse fissures, also raised by Mr. Dudley, we agree is one requiring further study for their explanation and elimination. The Bureau of Standards has begun an experimental investigation of this subject—following in part suggestions from Mr. Dudley.

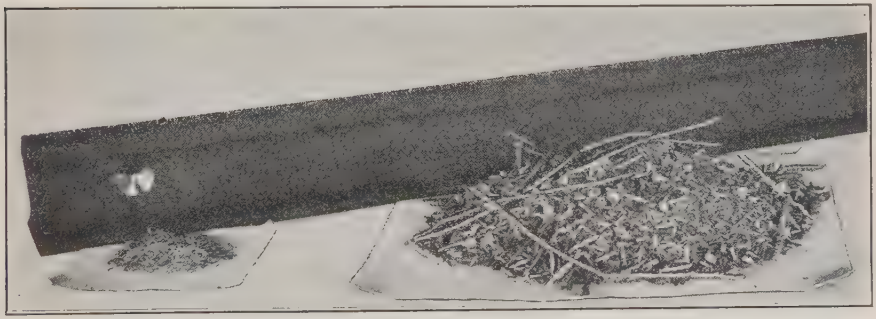
Manganese-Steel Rails

BY SIR ROBERT HADFIELD,* F.R.S., LONDON, ENGLAND

(New York Meeting, February, 1914)

SINCE the writer has been intimately connected with the development of manganese steel for many years, some remarks upon the early work with regard to the rolling and forging of this material might be of interest, and particularly as to the question of rails of special quality and of superior wearing qualities.

In a paper on Iron Alloys, with Special Reference to Manganese Steel, read before this Institute at the International Engineering Congress in Chicago in 1893,[1] the writer mentioned that, about 20 years ago, his firm in Sheffield produced railroad material, such as tires and axles, of forged and rolled manganese steel, thus showing that this product could be manipulated and worked up into the various required forms. The tires



Manganese Steel. Ordinary Carbon Steel.

11 g. removed from manganese-steel rail by drilling in 20 min., as compared with 750 g. from an ordinary carbon-steel rail in the same time.

FIG. 1.—MANGANESE-STEEL RAIL OF 7½ KG. PER METER ROLLED IN FRANCE IN 1904.

illustrated in the paper referred to above formed a particularly difficult class of work to produce in a special steel, but no special obstacles were met with. Although there was no difficulty at that time in making and producing manganese steel in various forms, forged, rolled or pressed; nevertheless, on account of its higher expense, sufficient encouragement

* Honorary member.

[1] *Trans.*, xxiii, 148 (1893).

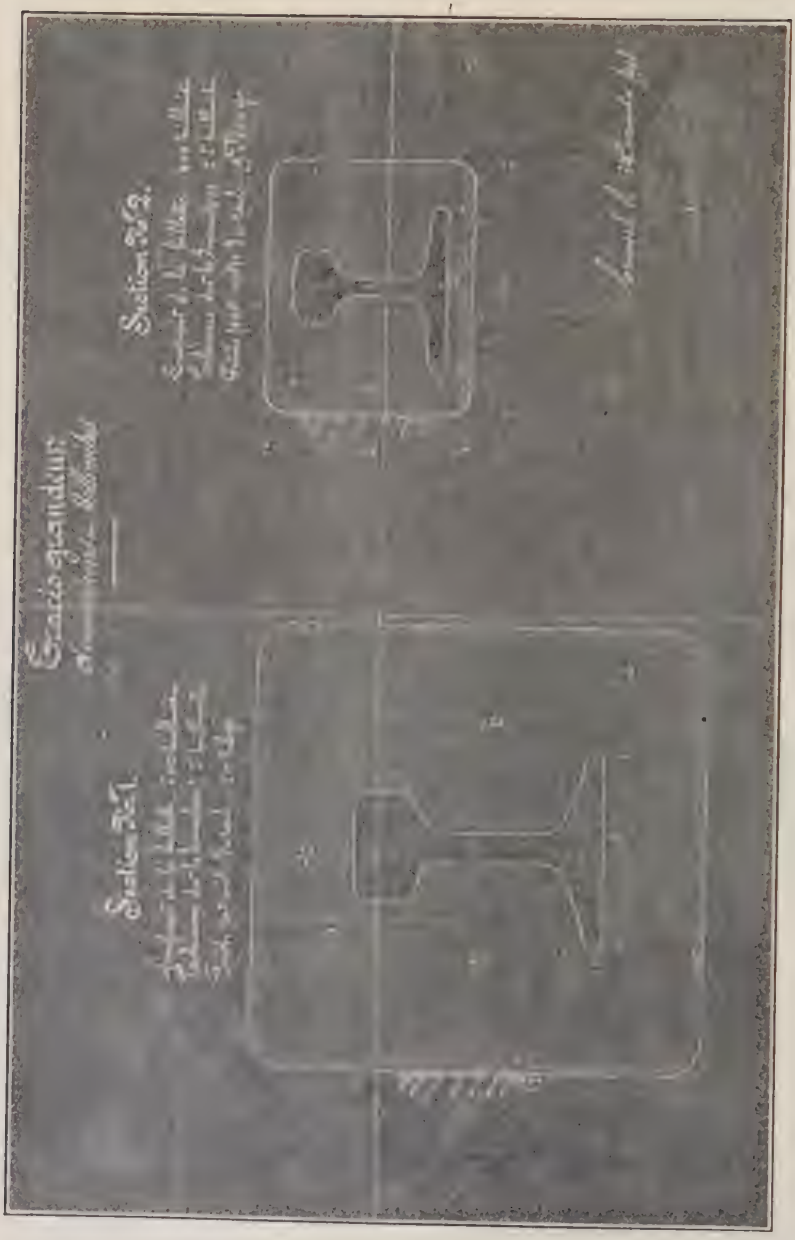


FIG. 2.—PHOTOGRAPH OF ORIGINAL TRACING PREPARED FOR THE PRODUCTION OF THE RAIL SHOWN IN FIG. 1.

was not then obtained from the users to introduce these products on a large scale. There never has been any real difficulty in obtaining forged, rolled, or pressed manganese steel, provided the consumer was willing to pay the necessarily higher cost as compared with ordinary steel.

The matter was further pursued in the present decade: In 1904, a French manufacturer produced rolled manganese-steel rails of $7\frac{1}{2}$ kg. per

FIG. 3.—SPECIMEN OF STEEL FROM WHICH RAIL SHOWN IN FIG. 1 WAS ROLLED. BENT DOUBLE COLD. TENACITY, 60 TONS PER SQUARE INCH. ELONGATION, 40 PER CENT.

meter, as shown in Fig. 1. The original tracing prepared by this manufacturer on Dec. 30, 1903, showing the sizes of the billets and rails to be produced, is reproduced in Fig. 2. These rails were rolled without any incident. One of them was experimented upon by the writer in the following manner: After drilling for 20 min., 11 g. of manganese steel were removed; while, in the same time, 750 g. were removed from an ordinary carbon-steel rail. The results of this test are shown in the photographic view, Fig. 1. A piece of this same manganese steel bent

double cold, as shown in Fig. 3. The material showed a tenacity of 60 tons per square inch, with an elongation of 40 per cent.

This test was so satisfactory that manganese-steel rails of heavy section, 100 lb. per yard, varying from 6 to 12 m. in length, were then produced in the same rolling mill. A section of one of these rails is shown in Fig. 4.

A large number of these rails were rolled in 1907, and were supplied to the Metropolitan Electric railway of Paris (in the commencing of operations on which, it will be remembered, there was some delay), and put to work, among other places, at the Bastile station in that city. These

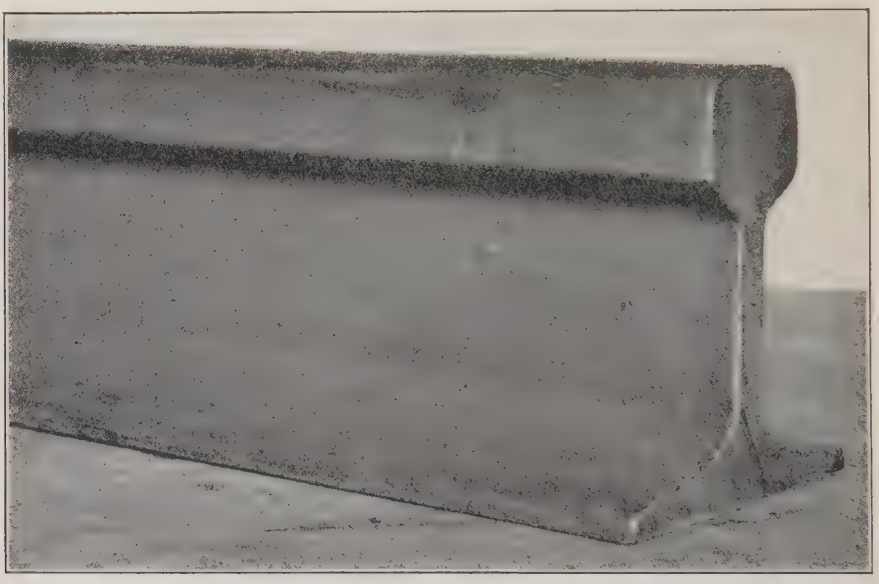


FIG. 4.—MANGANESE-STEEL RAIL, 100-LB. SECTION, TAKEN FROM A LOT OF 50 TONS ROLLED IN 1907 FOR METROPOLITAN ELECTRIC RAILWAY, PARIS, FRANCE.

rails, after several years of service on severe curve work, have given great satisfaction. After three years' hard service, the actual wear was found to be not more than about $2\frac{1}{2}$ mm. It has been estimated that the rails will remain in service for six to seven years before being worn out, whereas ordinary steel rails wear out and have to be replaced in less than a year. The rails in position are shown in Fig. 5.

A plant for producing sound ingots under the system devised by the writer was described in a paper[2] presented to the Institute at the February, 1913, meeting This system, as adapted to the manufacture of manganese-steel ingots, is illustrated in Figs. 6, 7, and 8. No less than 90 per

[2] *Trans.*, xlv, 473 (1913).

FIG. 5.—MANGANESE-STEEL RAILS, 100-LB. SECTION, ABOUT 40 FT. LONG, IN PLACE AT BASTILE STATION, PARIS, FRANCE.

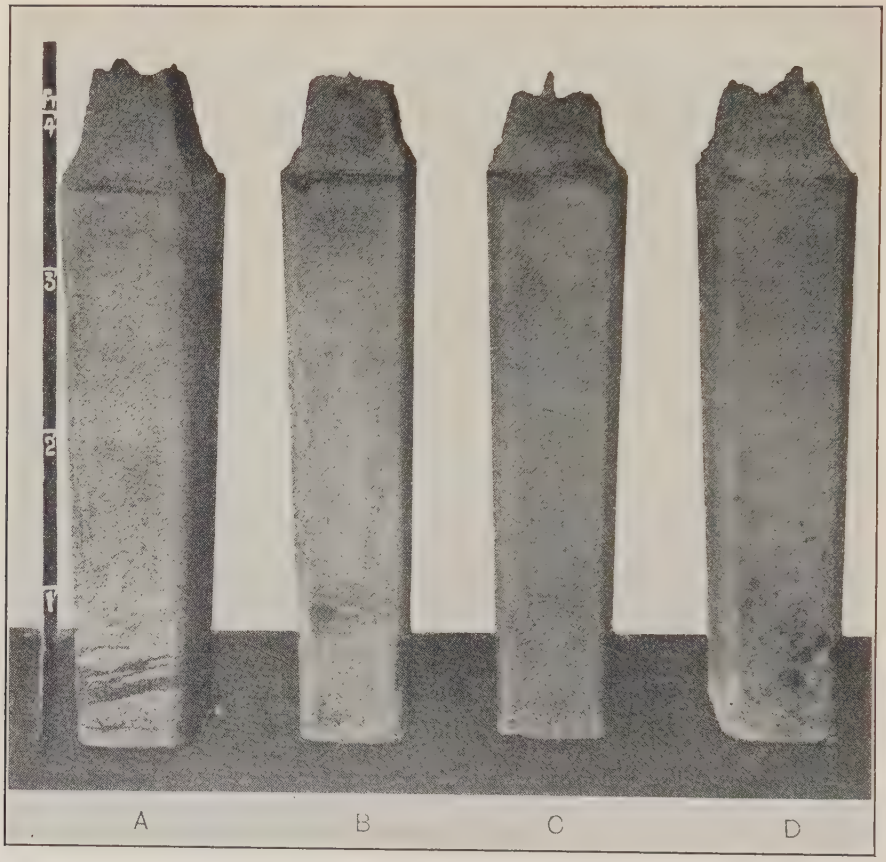


FIG. 6.—MANGANESE-STEEL INGOTS.

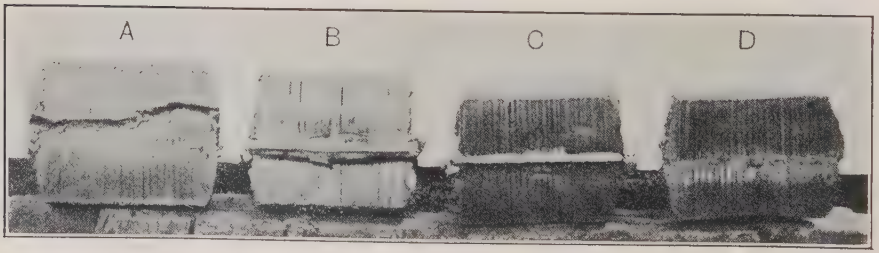


FIG. 7.—HEADS FROM ABOVE INGOTS PARTLY COGGED AND THEN CUT OFF.

cent. of sound material is obtained from an alloy steel, which will therefore be seen to offer great advantages Each 1 per cent. of expensive material saved means that much reduction in the cost of production.

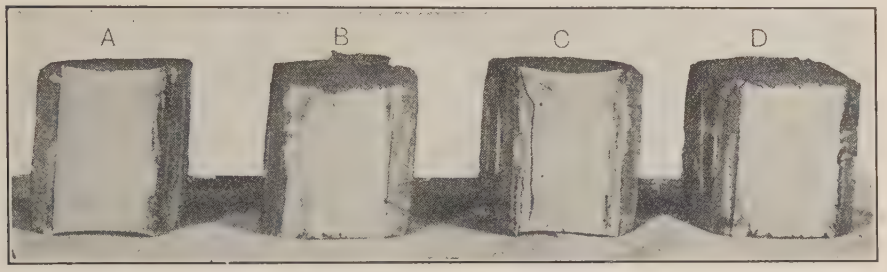


FIG. 8.—FRACTURES FROM THE UPPER END OF TOP BILLET FROM EACH OF THE INGOTS SHOWN IN FIG. 6.

The consumer has now to pay for 15, 20, or even 30 per cent. of waste in special steels made in the ordinary way, whereas this waste can to a large extent be avoided. Moreover, better and more uniform quality of alloy steel can be obtained.

DISCUSSION

W. S. POTTER, Pittsburgh, Pa.—The second page of this paper contains the repetition of the substance occurring at the bottom of the first page, *i.e.*, "There never has been any real difficulty in obtaining forged, rolled, or pressed manganese steel."

The writer refers to manganese-steel rails made by the Schneider Co. for the Metropolitan Electric Railway of Paris. In the *Revue de Métallurgie,* Paris, 1909, vol. vi, p. 550, in a letter dated Feb. 6, 1909, the Schneider Co. makes the following statements concerning the history of its efforts in the manufacture of manganese-steel rails:

"Our first tests made in this direction go back to March, 1904, and consisted in the rolling of some rails of 7.5 kg. per meter, obtained by the transformation of billets forged at the drop hammer. These tests were continued in February, 1905, by rolling, under the same conditions, two rails of 18 kg. to the meter.

"In August, 1906, and in anticipation of a furnishing of rails to the Metropolitan of Paris, we proceeded to a preliminary rolling of rails of 20.35 kg., which permitted us to recognize in advance the difficulties of manufacture which we would encounter, and to take all suitable measures for surmounting them. Six rails of 7 m. were thus rolled. The operation presented this peculiarity, that the transformation of the ingots into rails was effected exclusively by rolling (without preliminary forging of the ingots). Finally, from March to July, 1907, we proceeded under excellent conditions, to the rolling of 60 tons of Vignole rails, of 52 kg. to the meter, from 6 to 12 m. in length, at present in service at the Metropolitan Railroad of Paris."

On p. 560 of this journal the Schneider Co. makes the following statement:

"Manganese steel may be forged and rolled. Nevertheless, the quite special pains which its heating demands, as well as the very narrow limits of temperature between which one is restricted in working it, render these operations, the rolling of rails of great lengths especially, particularly difficult.

"We have, however, succeeded in rolling, in July, 1907, an important tonnage of Vignole rails, at present in service on the Metropolitan Lines of Paris."

The makers of the rails referred to by Sir Robert Hadfield differ radically from him as to their experience in the heating of manganese steel for rolling or forging. Until 1906, and after two years' experimenting, they were still unable to roll the manganese ingots without first forging them. Thereafter they found that the heating required "quite special pains" and that the metal could only be heated within "very narrow limits of temperature between which one is restricted in working it."

Personally, I have had 14 years' experience in the manufacture of cast manganese steel, rolled and forged manganese steel, and it would certainly be absurd for me to say that I had found "no difficulty," particularly in the heating of the cast metal for its conversion into rolled or forged forms.

Nine important steel companies in the United States, one in France, and two in England have wrestled with the heating of the cast manganese steel in their efforts to devise a practical method for its conversion into the wrought state. Hundreds, if not thousands, of men on both sides of the water will agree with me when I say that the preparation of the manganese-steel ingot for rolling is the very name of trouble. This art is entirely the opposite of what one would understand from reading Sir Robert Hadfield's paper submitted to-day to this Institute.

The first bar of rolled manganese steel which I ever saw was made by the Edgar Allen Co., of Sheffield, England, and was given to me for analysis when I was chemist of the William Wharton, Jr., Co., of Philadelphia. I found manganese 12.6 per cent. It was a very fine-grained, smooth bar, about 3 in. wide and perhaps $\frac{3}{8}$ in. thick. This was in the early part of 1901.

During the period 1901 to 1909 (more or less) many good men on both sides of the water struggled with the difficult matter of devising a method of heating an ingot of manganese steel in such a manner that it might then be rolled so as to afford a commercially available product. Outside of England and this country, no important progress in this art has been recorded. In Sheffield, England, the Edgar Allen Co. worked out a method, and in accordance with it rolled more rails than have ever been made by any company outside of the United States.

A representative of the Edgar Allen Co. described the successful Sheffield practice as follows:

"You 'eat it just so 'ot. Then you rowl it to the center, rowl it to the center, you understand. If you 'eat it just a little bit too 'ot, you crush your —— ingot. If you don't 'eat it quite 'ot enough, you break your —— rowls."

In other words, as per the written word from the Schneider Co., the temperature range as per the English practice, as developed in Sheffield and transmitted to the works of the Schneider Co., covered "very narrow limits of temperature between which one is restricted in working it."

During the same years, that is to say, from 1901 to 1907, inclusive, there was developed in this country a method of heating manganese-steel ingots for rolling, which, instead of being limited to a very narrow range of temperature (which according to the best information as to the Sheffield practice was from 850° to 950° C.), enabled the operator to go to 1,150° to 1,250° C. in the heating of the ingots. The result was a soft, ductile ingot, corresponding with a 20 per cent. carbon-steel ingot ductility.

In 1912 the Edgar Allen Co. was still selling manganese-steel rails at approximately $250 per ton. In 1913 they took license under my English patents and immediately reduced their price on rolled manganese-steel rails to approximately $125 per ton. The American price for several years has been $80 per ton.

J. W. Richards, So. Bethlehem, Pa.:—We probably all know that Mr. Potter has done more work than any one else in the United States on the rolling of manganese steel, and we all know, also, that Mr. Hadfield was the pioneer in the production of the metal, and that he certainly did forge and roll it a great many years ago. But the reading of his paper seems to confirm Mr. Potter's statement that he minimizes the difficulties, and as the proof of the pudding is in the eating thereof, the fact that by Mr. Potter's method manganese-steel rails are now being rolled and sold at less than one-half the price at which they were sold before his process came into use, would show that his methods are essentially a great improvement upon any previously known method of rolling this material

Sir Robert Hadfield (communication to the Secretary*).—With regard to Mr. Potter's remarks, I notice that he quotes a sentence of mine in which it is stated "There never has been any real difficulty in obtaining forged, rolled, or pressed manganese steel." As set forth in my paper, I again state that we in England have been rolling, forging, and pressing manganese steel as far back as the years 1887 to 1892, and have not found any special difficulties provided the ingots were made in a proper manner. Soon after the years referred to, among other articles, large quantities of dredger pins, all of which were forged, and of quite large diameter, 3½ to 4 in., were produced. This was referred to in my paper to this Institute in 1893; that is, more than 20 years ago. In view of the fact that it was then possible for us to produce forgings and bars from ingots of small and large sizes, it will be seen that my assertion is thoroughly confirmed by the facts.

According to recent statements in the technical press, it has only

* Received Apr. 16, 1914.

just been recognized by those new to this art that the question of making a suitable and satisfactory ingot is one of the chief points in connection with the successful production of forged and rolled manganese steel, a point which we in Sheffield have had in view during the last 20 years.

As regards Messrs. Schneider's remark that manganese steel could be readily forged and rolled, but that it requires care in manipulation, whether as regards heating or other portions of the process, this applies as much to-day as when the statement was made in 1907–1909. It will be admitted by all who deal with manganese steel that narrower limits of temperatures and methods of handling must be employed in its working. There is less latitude as compared with ordinary carbon steel, and it is this which is the gist of the statement very correctly put forward by Messrs. Schneider, especially as they had in view the production of rails up to 12 m. in length, the accomplishment of which is even to-day not a matter of the greatest ease. This is so not merely on account of its being a rolling-mill question, but in connection with other portions of the work which is necessary to be carried out in the manufacture of these rails. There is therefore nothing in the statement made by Messrs. Schneider to which exception can be taken.

In any case, however, as Messrs. Schneider definitely pointed out, they rolled manganese-steel rails in March, 1904, and in August, 1906, they rolled about 20 tons of these rails, of heavy section, 7 m. in length. Then, a few months afterward, 60 tons of manganese-steel rails were rolled, of heavy Vignole section, weighing 52 kg. per meter, and from 6 to no less than 12 m. in length. These rails are still in service, and have given exceedingly satisfactory results, as referred to in my paper. As these were probably the first manganese-steel rails of large section ever put into service with satisfactory results, full credit should be given to Messrs. Schneider. Therefore there seems to be nothing special in the claims they make in the remarks quoted which might not at that time have been made by any one dealing with manganese steel in its then new application for rails.

As is well known by those accustomed to dealing with manganese steel, one of the chief secrets of success is the careful handling during the first stages of the forging or rolling treatment. In other words, as Mr. Potter probably knows, if manganese-steel ingots are heated to temperatures such as those to which he refers, 1,150° to 1,250° C., unless there is careful handling in the rolls, the results will be disastrous. For example, if the rolls are too severe in their draft, then, apart from the question of temperature or the manner in which manganese steel is heated, it will not be easy to obtain satisfactory product.

It is of the highest importance that manganese steel when put to work should be perfect as regards its toughness, otherwise most disastrous results may ensue. This is referred to because some of the manganese-steel

rails produced in this country are stated not to have been satisfactory. This is mentioned in the American Railway Engineering Association's report for July last year, where it is stated that a number of rails put into use were while in service discovered to be broken. This is another proof that unless the manufacture is carried out very carefully, a point I have always particularly insisted upon, the use of this peculiar material, manganese steel, may prove dangerous. As is well known, manganese steel in its brittle condition breaks much more readily than carbon steel. It is also easy to have intermediate conditions in which under ordinary tests the material appears to be tough, yet, owing to imperfect heating and toughening treatments, it has a dangerous propensity to break with little warning. Another point may be emphasized: namely, that if the toughening of manganese steel is carried on at too wide a range in temperature, the risk is run of serious breakages occurring sooner or later. In such case the material would be blamed, when it is not really at fault, but the incorrect methods of treating it. These facts have in the past been fully and specially emphasized in my various papers and other communications.

With regard to the further remarks of Mr. Potter that there have been difficulties to overcome in dealing with this peculiar steel, this statement of course is quite correct. Others among us interested in the forging and rolling of manganese steel have at times not felt very happy in the past when visiting the scrap heap! In view of the facts previously referred to, it must be admitted that the problem was solved in England more than 20 years ago; and as regards suitable methods for producing rails, these were fully set forth to the metallurgical world in 1903, as a result of experimental work which had been going on for some time previous to that year. For example, attention may be called to the fact that in a recent technical publication[3] it is stated that the method by which manganese-steel rails can be satisfactorily produced is "by using inverted molds (that is, ingots cast with the large end up and the small end at the bottom) with refractory tops, putting carbon on the top of the ingots after pouring, and keeping this in a state of incandescence with compressed air." If this system was not employed, the maker, it is stated, met with innumerable difficulties in handling the fluid steel, or, to use the language of the article referred to, before using this device "We were making some of the most beautifully piped ingots you ever saw." Methods of this nature and similar to those above described have been urged by me ever since 1903, and full details have been set forth in my papers and other communications. Under the system my firm has made, in a most satisfactory manner, very large tonnages of perfectly sound manganese-steel ingots, and showing little if any more waste than ordinary carbon steel.

[3] Troubles with Manganese Steel: *Iron Age*, vol. xciii, No. 7, p. 447 (Feb. 12, 1914).

Under the same system are produced carbon-steel ingots absolutely sound and free from piping.

It is therefore no little satisfaction to find that if the article referred to in *Iron Age* correctly represents the facts, as no doubt it does, the successful production of manganese-steel rails in this country is being based upon the system described so many years ago. If this system is a success for producing ingots from which manganese-steel rails are made, may I not urge that it also possesses the same advantages for ordinary steel ingots from which carbon-steel rails are made.

With reference to the statement that "Nine important steel companies in the United States, one in France, and two in England have wrestled with the heating of cast manganese steel," I can only repeat that long before the date mentioned, 1901, we had wrestled, and this most successfully, with the art of producing rolled and forged products of this steel. It will be seen, therefore, that it is not possible to agree with the remarks made under this head, and it seems to me that others, in the light of the above-mentioned facts, will take the same point of view.

Reference is made to the efforts of those who were not the inventors of manganese steel, and therefore could not have been the first to produce it. It was probably not until some 10 or 12 years after my company produced this material in rolled, forged, pressed, and other worked forms, that the firms referred to produced the material which I discovered and first worked at between the years 1883 and 1887, before announcing to the world in my papers this discovery of mine. Therefore, at least 15 years before 1901, the date when Mr. Potter says he first saw a bar of rolled manganese steel, my firm had been doing this, which was naturally so, seeing that I discovered the material. Moreover, in view of my papers to the Institution of Civil Engineers in 1888, and later to this Institute in 1893, and the references therein to the many tons of rolled bars, sheets, axles, tires, forgings and other forms of manganese steel, I am at a loss to understand why in the remarks referred to these evidences are overlooked.

With regard to the temperatures used for forging and rolling, there is full and definite evidence on record that the early practice of dealing with manganese-steel ingots from the year 1887 and onward was to heat the ingots very much hotter than the temperatures to which Mr. Potter refers. He seems to have got an erroneous impression both with regard to the large quantities and as to the manner we in England have been rolling, forging and pressing manganese steel during the last 20 years. I should be very glad to show him our experimental records, and am sure he would then see that many of the suggestions he makes were anticipated by our early work. He would also then see the very large

tonnages of forged manganese-steel work we have turned out from the year 1887 and onward.

I should like to point out, with reference to the remarks of Professor Richards, that there seems to be some misunderstanding about the question of the work previously done with regard to reducing manganese steel by forging and rolling from ingots. May I once more emphasize that for more than 20 years rolling, forging, pressing, and otherwise manipulating manganese steel in forms other than those cast has been carried out on a large scale. Long before the year 1900 many thousands of tons of such forged and rolled products were manufactured and used. The methods of carrying out such work—that is, the temperatures employed for rolling and forging—are practically the same to-day, 1914, as at the end of the last century.

Moreover, in the paper presented to this Institute by the writer in 1893, entitled Iron Alloys, with Special Reference to Manganese Steel, full particulars and photographs are shown of manganese steel (*a*) forged into railroad axles, (*b*) rolled into railroad tires; also many other worked forms of this material.

As regards the selling price, that is quite another matter. There were in former years many reasons why the selling price should not have been reduced, but the cost—and this is the important point— of working manganese-steel ingots into, say, billets, blooms, bars, or forgings was, as just mentioned, pretty much the same many years ago as now, excepting of course improved machinery, such as more powerful rolling mills, presses, etc., effecting diminution in cost; this, however, would not make a large reduction in the cost. It would be quite possible to show the original costs of working manganese steel and compare them with to-day's figures, in order to show that the above statements are correct; in fact, owing to higher wages and more costly fuel, probably there would be a slight increase in the cost.

The American Steel-Rail Situation

BY ROBERT W. HUNT, CHICAGO, ILL.

(New York Meeting, February, 1914)

One of the most serious and important economic administrative problems facing American railway authorities to-day is that of their rails, and it is one to which much thought is being given, not only by the executive officers of the railways and the manufacturers of the steel rails, but also by State and National commissions. It is realized by many and admitted by some that the present situation is one which cannot continue. The roadways of railways must be made more secure, or the weight of rolling stock and the speed of trains must be lessened; and the desired result must be attained with the minimum outlay of money, both in expenditure on plant and in cost of operation. If the investment per mile is too large, or through the lessening of tonnage and efficiency of equipment the cost of operation is too great, the desired and demanded cheap service cannot be rendered to the public. Therefore, the weight of equipment cannot be greatly reduced, and lessening the rapidity of service would be far from satisfactory; consequently, the safety of the roadway must be secured with the least practical outlay.

It is imperative that the roadbed should be properly graded, with the fewest practically possible curves; that it should have good ballast, be well drained, have good ties properly spaced, and be laid with sound rails of suitable weight, efficiently joined together, and that the whole property be carefully and intelligently cared for.

My province is to treat of the sound rails.

During the last few years there has been renewed interest in and discussion of that question, and I know that many rail makers not only have been and are desirous of maintaining the highest standards of manufacture which they may have attained, but have been and are seeking to better their product, in several instances devoting much time and money to that end. At the same time, they realize that the cost of any improvement in quality which they may accomplish must be kept within certain commercial bounds; and that fact is as fully understood by their patrons as by themselves, and it should be kept in mind by all who discuss the question.

As a matter of record, I present in Fig. 1 a summarized statement of what may be called the mechanical practice of the steel-rail mills of the United States and Canada.

It will be noted that the several works differ in their practice as to kind of steel, size of heats, size of ingots, kind of blooming and rail trains, number of passes in the rolls of both trains, and as to direct rolling into rails; also as to the details of the finishing departments of the several mills. In consequence of such variations, there are, of course, differences in the operating practice of the several plants. One thing that, in my judgment, has had a great effect upon the quality of the product of all the mills is that the workmen have been and are paid on a tonnage or piece basis, with, in some cases, an additional prospective bonus based on quantity of product. Unfortunately, after the produced rails had left the works, there was but little chance of the identity or individuality of the workmen in the different departments of the works, who made them, being connected with them. It is true that a number corresponding with that of the heat of steel from which they had been rolled, and the month and year in which they were made, and the name of the works, were branded on each rail, but to actually identify the steel maker who made the steel, the heater who heated it, the roller who rolled it, the shearman who cropped the blooms, etc., would have been a complicated and practically impossible proposition. One result was that if, for any reason, such as delays from accidents to machinery, etc., the quantity of product was threatened, there was temptation to in some way cut corners, the workmen knowing that if the rails were once out of the mill they need not worry over any individual responsibility, which feeling was simply human.

In an endeavor to meet this and other phases of the situation, some two years ago I ventured the establishment of a system of more constant and thorough inspection of rail making than had been generally, if at all, practiced. In this movement I received the support at first of several and later of many of the railroad companies of this continent, and I am happy to say that I have also received the loyal support of the officials of the several steel companies. This system of inspection I explained in a former paper, presented at the Cleveland meeting, October, 1912.[1]

I have repeatedly shown that without sound ingots there will not be any certainty of producing sound rails; and that, as each ingot is an individual steel casting, some means should be adopted of testing for soundness the rails produced from each ingot; and until reliable ways of casting sound ingots are adopted the importance of such testing should not be overlooked or underestimated. It is possible to make sound ingots. It will cost more than the present practices; but is that a good argument against it? Whether that increased cost should be added to the selling price of rails is a commercial question outside of the province of this paper. The increased cost of individual ingot testing is so small that it should not be a matter of much commercial importance.

[1] Recent Developments in the Inspection of Steel Rails, *Trans.*, xliv, 269 (1912).

AMERICAN STEEL RAIL MILL PRACTICE

COMPILED BY ROBT W. HUNT & CO. CHICAGO, ILL.

OCTOBER 1ST 1913

C. W. Gennet, Jr.

NAME OF COMPANY	NAME OF WORKS	LOCATION	NO. OF BLAST FURNACES	RATED PRODUCTION OF EACH IN 24 HRS (TONS)	BESSEMER CONVERTERS: NO	BESSEMER CONVERTERS: RATED CAPACITY EACH (TONS)	OPEN HEARTH FURNACES: NO	OPEN HEARTH FURNACES: RATED CAPACITY EACH (TONS)	KIND OF STEEL MADE FOR RAILS	NOMINAL SIZE OF INGOT MOLDS (INCHES): SECT AT BOTTOM	NOMINAL SIZE OF INGOT MOLDS (INCHES): HEIGHT	NOMINAL NO. OF INGOTS PER HEAT: BESS.	NOMINAL NO. OF INGOTS PER HEAT: O.H.	NOMINAL SOAKING PIT CAPACITY: NO. OF INGOTS	TYPE OF BLOOMING MILL (d)	NO. OF PASSES GIVEN TO INGOT IN BLOOMING MILL	NOMINAL SECTIONAL SIZE OF BLOOM MADE: INCHES	BLOOMS RE-HEATED	NO. OF PASSES IN RAIL MILL	NOMINAL NO. OF PASSES INGOT TO RAIL: MIN.	NOMINAL NO. OF PASSES INGOT TO RAIL: MAX.	NO. OF HOT SAWS GIVING USED	DISTANCE NEAREST HOT SAW FROM FINISHING PASS: FEET	AVERAGE TIME(SEC) OCCUPIED LEAVING FIN PASS & BEING SAWED	BRAND ON TOP OR BOTTOM SIDE OF RAIL AS ROLLED	BRAND ON RAIL READS FROM — TO — OF INGOT AS ROLLED	FOR 100# RAIL: NO. OF BLOOMS CUT PER INGOT	FOR 100# RAIL: NO. RAILS MADE FROM EACH BLOOM	FOR 100# RAIL: TOTAL NO. RAILS PER INGOT	FOR 100# RAIL: NOMINAL WEIGHT PER SS RAIL (POUNDS)	FOR 90# RAIL: NO. OF BLOOMS CUT PER INGOT	FOR 90# RAIL: NO. RAILS MADE FROM EACH BLOOM	FOR 90# RAIL: TOTAL NO. RAILS PER INGOT	FOR 90# RAIL: NOMINAL WEIGHT PER SS RAIL (POUNDS)	FOR 80# RAIL: NO. OF BLOOMS CUT PER INGOT	FOR 80# RAIL: NO. RAILS MADE FROM EACH BLOOM	FOR 80# RAIL: TOTAL NO. RAILS PER INGOT	FOR 80# RAIL: NOMINAL WEIGHT PER SS RAIL (POUNDS)	NO. OF HOT BEDS USED	ARE RAILS FLOOR-CHILLED OVER CORE?	NO. OF RAILS PER INGOT	ARE RAILS PUT UP SYMMETRICALLY?	NO. OF COLD STRAIGHTENING MACHINES	NO. OF SETS OF DRILLS
Algoma Steel Co.	Algoma	Sault Ste Marie Ont. Can.	3	2-300 1-400	2	5	5	35	Bess. & O.H	18×19	72	2	18	96	2 High Reversing	15-17	8×8	Yes	11	26	28	1	120	30	Bottom	Top	Bot.	2	2	4	6¼					2	3in1 2in1	5	6¼	2	Yes	65	Head	5	5
Bethlehem Steel Co.	Saucon	So. Bethlehem, Pa.	6	2-200 4-450	2	25	10 6	50 75	Open Hearth (straight & Duplex)	19×23	72		24	120	2 High Reversing	13-15	8×8	No	11	24	26	1	211	33	"	Bot.	Top	2	3	6	6½	2	4in1 3in1	7	6½	2	4	8	6½	6	"	90	Base	6	6
Cambria Steel Co.	Cambria	Johnstown, Pa.	8	4-400 4-300	4	12	28	50-100	Bess. & O.H.	20-23 23×23	78 78	4	16-24	140 1 Mill 180 1 Mill	3- 1- 2 High Rev. 3 High	14-28	9×10	Yes	12	26	40	4	79	30	"	May be either		Various depending on size of Ingot										4th	"	70	Head	8	4
Carnegie Steel Co.	Edgar Thomson	Braddock, Pa.	11	500	4	15	14	75	Bess. & O.H.	18½×19½	71	7	40	160	3 High	7	9½×9½	Yes	13	20	20	4	43	15	"	Bot	Top	2	2in1 1in1	3	6½	1	3	3	6½	2	2	4	6½	4	"	85	Base	12	12
Carnegie Steel Co.	Ohio	Youngstown, Pa.	6	500	2	12	12	75	Open Hearth	19×21	74		26	144	3 High	9	8×8	No	11	20	20	3	122	25	"	Bot.	Top	2	2	4	6½	2	2	4	6½	2	3in1 2in1	5	6½	2	"	70	Base	6	6
Colorado Fuel & Iron Co.	Minnequa	Pueblo, Col.	6	2-250 4-425	2	10	12	65	Open Hearth	18×20	67		30	144	2 High Reversing	13-17	8×8	No (k)	10	23	27	1	108	26	"	Bot	Top	2	2	4	7	2	2	4	6½	2	2	4	6½	3	"	60	Head	6	6
Dominion Iron & Steel Co.	Dominion	Sydney N.S. Can.	6	250	3	18	10	50	Open Hearth (straight or Duplex)	18½×21½	82		18	90	2 High Reversing	13-19	8×8	No (k)	11	24	30	1	107	25	"	Top	Bot.	2	2in1 3in1	5	6½	2	2in1 3 in1	5	6¾	2	3	6	6¾	2	"	170	Head	4	4
Illinois Steel Co.	Gary	Gary, Ind.	8	400	None	None	28 4	80 100	Open Hearth	20×24	79		24	192	4 Sets - 2 High 1 Set - 3 "	9	8×8	No	9	18	18	5	158	29	"	Bot	Top	2	3	6	6½	2	3in1 4in1	7	6½	2	4	8	6½	6	"	80	Base	16	16
Illinois Steel Co.	South	So. Chicago, Ill.	11	500	3	15	26	50	Bessemer (or Electric)	18×19½	67	7		168	3 High	9	8×8	No	9	18	18	5	60	25	"	Bot	Top	1	3	3	6½	1	4	4	6½	1	4	4	6½	3	"	90	Head	16	16
Lackawanna Steel Co	Lackawanna	Buffalo, N.Y.	7	500	4	15	14 2	60-90 200 Tilt	Bess. & O.H. (straight or Duplex)	19×19	72	6	34	120	2 Sets - 2 High 1 Set 2 High Rev.	6	8×8	No	9	15	15	1	270	24	"	Top	Bot.	1	4	4	6½	1	4	4	6½	1	5	5	6½	4	"	120	Both	12	12
Maryland Steel Co.	Maryland	Sparrows Pt. Md.	4	300	3	15	5	60	Bess. & O.H. (straight or Duplex)	20×21	68	6	22	108	2 High Reversing	13-17	8×8	No	11	24	28	1	158	32	"	Bot	Top	2	2	4	6½	2	2in1 3in1	5	6½	2	2in1 3in1	5	6½	6	No	150	Head	7	7
Monterey Iron & Steel Co.	Monterey	Monterey, Mexico	1	350	1	10	4	30	Open Hearth	15×22	66		16	72	2 High Reversing	15	8×8	Yes	10	25	25	1	205	40	"	Top	Bot.													2	Yes	60	Head	3	3
National Tube Co.	Lorain	Lorain, Ohio	5	500	2	15	6	80	Open Hearth	22×26	72		22	64	2 High Reversing	21-25	Various (m)	Yes	11 (f)	32	36	2	123	26	Top	Bot	Top	3	2	6	6½	3	2	6	6½	4	2	8	6¾	4	No	120	Both	12 (j)	11
Pennsylvania Steel Co.	Pennsylvania	Steelton, Penn.	5	2-200 2-460 1-500	3	15	Basic Acid	75 85 85	Bess. & O.H. (straight & Duplex)	19×19	74		30	60	3 High	9	Various (m)	Yes	11 (f)	20	20	2	8	24	"	Top	Bot	2	2	4	6½	2	2	4	6½	3	2	6	6½	4	Yes	70	Both	4	6
Tennessee Coal Iron & R.R. Co.	Ensley	Ensley, Ala.	6	350	2	25	8	100	Open Hearth (Duplex)	23×23	79		30	144	2 High Reversing	11-15	8×8	No (k)	10	21	25	4	120	25	Bot.	Top	Bot					3	3in2 2in1	8	6½	3	3	9	6½	4	Yes	75	Head	10	10

NOTES

a – Including those in process of Construction

b – Continuous Tube Process furnaces giving 100 tonners

c – Also one Heroult Electric Furnace of 15 Tons capacity

d – Unless otherwise stated Mills are non-reversing and have fixed distance between Rolls.

e – Varies according to Section of Rail being rolled

f – For Standard T Rails

g – Or one commonly used

h – Not protected by roof from weather

j – Including 4 Automatic Machines

k – Some Blooms may be re heated.

FIG. 1.—SUMMARIZED STATEMENT OF PRACTICE IN RAIL MILLS OF THE UNITED STATES AND CANADA.

There is another feature of the situation which demands and is receiving serious consideration from railway officers, consulting engineers, the steel works officials, and others. That is, the desirability, if not the absolute necessity, of increasing the weight of rail sections. A number of such sections have been designed, and some have been rolled, and the rails are in use; but there is a metallurgical feature involved which should not be overlooked.

The experience of many railways has been that their earlier rails, which were of lighter sections, gave better service than the later heavier ones, and the track men will tell you that when they cut one of the old rails they found a close, fine-grained structure, while the larger rails show a more or less coarse one. The old lighter rails permitted the fining effects of the rolling to penetrate, and, the mass of metal in their heads being comparatively small, the effect of the interior contained heat passed off sooner. The foregoing conditions result in the webs and flanges of all rails having a finer structure than their heads; therefore, as we increase the size of the rail sections we will certainly decrease their proportionate strength, and under present manufacturing conditions the heavier rails will have less resistance to the abrasive wear of the traffic. Nevertheless, there seems to be a necessity for heavier rails, and so much the better, if we can also make them proportionately stronger.

Excepting in the ability to roll rails by the use of fewer passes in the rolls, and to handle the operations mechanically and automatically, there have not been any radical changes in rail rolling. The plan of rolling reductions has remained much the same. Some experiments have been made, and various schemes have been proposed, but none of them are in active use. In my judgment, the present situation demands serious consideration, even though it should require very radical changes in the rolling machinery of the existing rail mills. If we must have rails with more metal in their heads, as well as thicker webs and flanges, it is most important that the work of reduction and formation from the, say, 8 by 8 in. section of the blooms shall be applied in a way to penetrate and fine the metal in the rail heads. I shall not here attempt to particularize the way, but no doubt a modification of the universal mill, *vide* the Gray mill for structural sections, will suggest itself as one possible way, and I know of several schemes which contemplate putting work on the top of the heads in addition to only the sides, as is the case with the present mills. If that must be accomplished to solve the problem, it can and will be done.

Once more I emphasize the truth that the physical treatment of the metal is of at least equal importance with its chemical composition.

The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements

BY J. E. JOHNSON, JR., NEW YORK, N. Y.

(New York Meeting, February, 1914.)

At the Cleveland meeting of the Institute in October, 1912, I had the honor to present a paper outlining the conditions surrounding the charcoal iron industry, and giving some of the results of an investigation started about three years ago for the purpose of determining the reason for the poor reputation of the Lake Superior charcoal irons as compared with the Southern and Eastern irons.

In that paper I explained the effect of high carbon on quality, and tried to make clear that that paper was a preliminary one, as we felt convinced that we should eventually discover other factors affecting quality.

The progress of the investigation has brought out facts that seem to us of the utmost importance practically and theoretically, and the purpose of the present paper is to present these facts with the proofs which seem to us to establish them.

The closing year of the nineteenth century saw the publication of the first definite and connected statement of the relationship between the cast irons and the steels.[1] The metallurgical world was ready to accept the fundamental unity of these two series, acceptance being greatly hastened by Professor Howe's classic paper, The Constitution of Cast Iron, presented at the Richmond meeting of the Institute, February, 1901.

But the doctrine of chemical control of the quality of pig iron met some obstacles, small but exceedingly difficult of removal. The old

[1] See article, entitled The Chemistry and Physics of Cast-Iron Briefly Considered, by J. E. Johnson, Jr., published in the *American Machinist* of Apr. 5 and 12, 1900. In this article the continuity of the cast iron-steel series was set forth at some length and the relationships between soft cast iron on one side and soft steel on the other to white iron as the connecting link, were pointed out in detail, also the mechanical effect of the graphite in reducing the strength of the matrix, and many other matters now perfectly familiar to all.

school of metallurgists, who knew but little chemistry, had always insisted that there were fundamental differences in quality which were not revealed by analysis. The more ardent advocates of the newer theory of chemical control ridiculed this view. Those whose position forced them to look for the facts from whatever source derived came gradually to believe that while the theory of chemical control was in a broad way correct, there were certain facts which this theory did not cover, and that the old-school metallurgists, while wrong in many of their notions, were undoubtedly right as to this.

This view now seems to prevail among those best informed on the subject, and I shall not reiterate here the profusion of hard facts confirming it, which any one may find for himself who will leave the beaten track of foundry work and investigate those by-paths in which special qualities of product are made.

In our search for the explanation for these discrepancies, there seemed, after the most careful consideration, to be two facts which might be taken as established:

1. That certain irons possessed a quality as they came from the furnace which was preserved on remelting, and which reappeared in castings of which these irons formed an important ingredient.

2. That the rarer elements, such as nickel, cobalt, chromium, titanium, vanadium, and the like, while not absent from the point of view of ultimate analysis, were in such small quantity and varied in ways so entirely independent of the quality of the iron that the explanation could not be attributed to them.

In spite of the certainty in our own minds regarding conclusion 1, and the amount of qualitative confirmatory evidence we had on the subject from innumerable users, we felt that there might be some hesitation on the part of others in accepting it, and we accordingly decided to make a series of tests which should, if possible, be so conclusive as to admit of no dispute.

We took six irons constituting three pairs, the irons of each pair containing the same silicon (approximately) but having a very different fracture and different strength, as shown by test bars cast from the furnace. The silicon of the three pairs was about 1.90, 1.00, and 0.70 per cent. (See Table I., p. 354, for complete analysis.) We remelted these in twin crucibles, as will be described later, and cast test bars from the remelts.

In addition to the silicon we determined the graphitic and combined as well as the total carbon in each iron, both before and after remelting. We also made photomicrographs of the original pig and

of a test bar from the remelt, both etched and unetched, for each of the six irons. We have arranged these in groups of eight, each representing one of the pairs. These show at the left the photomicrographs of the original iron; at the right, those of the remelts. Above in each case are the good irons, below are the poor ones. Underneath the photographs are given the silicon, and the breaking strength of the 1.25-in. bars. All of the specimens were taken from corresponding places in the pig or test bar, and are fairly representative of the structure as a whole.

We believe that these data is this form will carry conviction to the minds of even the most unwilling, and that no one will ever be able, in the face of this evidence, to contend successfully that there is not some quality in iron which persists on remelting and is independent of its analysis for ordinary elements.

In order to show quantitatively the relation between strength, combined carbon, and graphite, we made combined carbon determinations, by difference in all cases, on the 1-in. bars, but the greater chilling of 1-in. bars in the Medium No. 3 iron threw the combined carbon far beyond the eutectoid ratio, and we did not think this constituted a fair comparison. We therefore ran the combined carbon in all the 2-in. bars, taking the sample from the exact center of the broken face of the 2-in. bar, so as to make the results as nearly comparable as possible in each case. These results are shown in Table I.

In pair 1, silicon 1.90 per cent., the combined carbon in the strong iron was 0.65 per cent., and in the weak iron 0.78 per cent. The graphite in the strong iron was 3.40 per cent., in the weak iron 3.30 per cent.

In pair 2, 1.00 per cent. silicon, the strong bar contained 0.96 per cent. combined carbon, the weak iron 1.13 per cent. The graphite was exactly the same in both, 3.00 per cent.

In pair 3, 0.70 per cent. silicon, the combined carbon in the strong iron was 1.20 per cent., in the weak iron 1.29 per cent. The graphite in the strong iron was 2.90 per cent., that in the weak iron 3.20 per cent.

In every one of these cases it will be seen that if we take account only of the quantity of combined carbon as increasing the strength of the matrix and that of graphite as reducing the same, we shall get results exactly opposite to the actual ones, except as regards the graphite in pair 3, and even here the difference in favor of the strong iron is slight. It may be that the matrix is somewhat stronger in the strong iron, but inspection of the three sets of photographs with an unprejudiced mind will force one to the conclusion that the principal

reason for the difference is the shape and size of the graphite formed in the two cases, and that quantitatively this difference is ample to account for the difference in strength.

It seemed inconceivable that a quality which would survive remelting and superheating to a point well above the fusion point, as in foundry practice, should be a purely physical quality. In other words, if one iron were better than another because cast at a lower temperature, for instance, and for no other reason (both being identical in analysis); when both were remelted to the same temperature and recast under the same conditions, it is incredible that they would not lose the difference in quality due to this difference in original casting temperature. As the difference in quality evidently did persist in spite of remelting, it seemed to me necessary to believe that the difference was a chemical one.

The Explanation to Be Sought in the Common Gaseous Elements

As a consequence of this conclusion and of the establishment to our satisfaction of the second fact above mentioned, that the less common elements were not responsible for these differences in quality in general (though undoubtedly they exert marked influence in special cases), it seemed inevitable that some of the common elements, not ordinarily determined, must be accountable for these variations in quality. Of these, the most universally present in the blast furnace, oxygen, nitrogen, and hydrogen, are the most common and seemed a likely field for investigation.

In regard to hydrogen, it was a very simple matter to test for its presence in the form of water vapor in the current of oxygen passing through the combustion furnace in carbon determinations, and accordingly precautions were taken to see if any water vapor could be detected in the oxygen current, but with an absolutely negative result. This, of course, was not surprising in view of the small quantity of hydrogen in the hearth of the furnace compared with the quantity of nitrogen and oxygen present, and in view of the results of recent investigations along this line. All of the latter prove that while hydrogen is absorbed by iron with avidity at low temperatures, it is all driven off on raising the temperature to a point much below that of the blast furnace.

There remained, therefore, as likely fields of investigation, oxygen and nitrogen.

Oxygen in Coke Irons

Some metallurgists have taken the view that it was impossible for oxygen to exist in the presence of so highly carbonized a product as cast iron, and at so high a temperature as that of the blast furnace, but about 10 years ago I had made certain observations which seemed to me to indicate strongly the presence of oxygen in cast iron, and I did all that I could to get the matter investigated then by those competent to do the work. I wrote to a number of distinguished chemists and metallurgists, told them what I had observed, and begged them to take the subject up and follow it to a conclusion, I not being a chemist myself and having no chemist in my employ at that time capable of working the matter out. Some of those to whom I wrote were politely indifferent and others mildly interested, but too busy on other things to concern themselves with such an investigation.

The facts observed at that time have a sufficient bearing on the present matter to justify their presentation here.

The furnaces of the Longdale Iron Co., in whose employ I was at that time, ran continuously for many years on basic iron, with an absolutely constant ore mixture and with a blast temperature which was, to all intents and purposes, unvarying from one year's end to another, 850° F. Careful and long continued observation of the appearance of the iron, and particularly of the small sample chill pig made at each cast, in conjunction with the analysis, taken with the working of the furnace itself and the character of the running iron, brought out clearly certain conditions which prevail to a certain extent at all plants, but not so conspicuously at those with less constant conditions.

Marked differences in the fracture of the iron for a given analysis were observable according to whether the iron was "wild" on irregular furnace conditions, or whether the operation was normal and the iron "quiet."

On white irons, in good normal working condition of the furnace, the fracture of the chill pig was practically square, with clearly marked acicular crystals running perpendicularly from the chilled surfaces of the pig, even from the top in some cases, and clearly showing lines running from the center to the corners, these lines being where the sets of crystals from adjacent sides met.

On a "wild" iron of similar analysis the fracture was smooth, but conchoidal rather than square, with a complete absence of crystals radiating from the sides of the pig, and, of course, a corresponding absence of the interference lines of these adjacent sets of crystals running to the corners.

Irons with higher silicon when made on a smoothly running furnace had a comparatively slight chill. Such as there was showed strongly marked acicular crystals similar to those in the white iron, the grain quite open and the color light. When made on a wild-working furnace, iron with approximately the same silicon had greater chill with absence of crystals, while the gray portion was exceedingly fine grained, and dark, almost purplish in color, so as to give it a velvety appearance.

In coke practice a wild-working furnace is almost always associated with higher sulphurs, and the irons last described, both white and gray, came to be cordially detested as showing, even in advance of analysis, high sulphur.

It is a matter of much interest to observe that the white irons showed in an increasing degree, as they became wilder, cleavage surfaces which were discolored with the blue, red, and purple "oxide colors." This was commonly attributed to oxidation by the absorption of air into cooling cracks in the pig, but I have seen many cases when these surfaces could only be broken apart by the most tremendous sledging, and this never seemed to me to correspond with the idea of a crack of sufficient width to permit the ingress of air.

These "wild" irons, particularly the harder ones, as they cooled, threw off a scale of considerable thickness, continuous at first and detaching itself from the surface of the pig in slabs frequently as large as one's hand. Thinking that this scale might be an indication of something, I had it analyzed for sulphur, with the surprising result of finding only about one-third as much sulphur in the scale as was in the iron.

After the chemist who made this analysis had left, I was discussing this remarkable result with his successor, who had been the assistant at that time. He told me that the analysis had been made by the evolution method, and that if the sulphur were in the oxidized condition it would not be given off by this method, causing a low result, and that it would be necessary to use the gravimetric or fusion method to obtain correct results in this case.

On the first subsequent occurrence of this scale I accordingly had an analysis made by the latter method, with the result of finding, instead of one-third as much sulphur as in the pig, about three times as much.

About this time Prof. E. D. Campbell, of Ann Arbor, published a paper describing the remarkable ability of the oxysulphides of iron and nickel to penetrate the solid walls of iron tubes. It then seemed

possible that this scale might be an oxysulphide, which exercised this peculiar penetrating power to make its escape through the crystals of the solidifying pig, and gathered on its surface.

If this were true, then the "wild" irons which showed this scale should contain some of their sulphur in the oxidized condition, and the results obtained by the ordinary volumetric analysis for sulphur would show a lower result in such irons than the gravimetric, whereas, in the irons on a normal working furnace (in the absence of titanium) a small and constant difference is found. Accordingly, it seemed that the difference between the volumetric and the gravimetric sulphur should increase from normal iron to "wild" iron.

Quite a number of comparative analyses were made to determine whether this were actually true, and the results confirmed this theory to a remarkable degree. Of something like 12 irons picked to show either a high or a low difference between volumetric and gravimetric sulphur, according to the way the furnace was working, about 10 showed the expected result. No record, unfortunately, has been preserved of this work, but I remember that in one case the gravimetric sulphur was 50 per cent. higher than the volumetric, the latter being 0.034 and the former 0.051 per cent., this being on a particularly "wild" iron.

Oxygen in Charcoal Irons

When I first began making charcoal iron, the founder told me of "special cylinder" iron, which he described as "an iron with a No. 2 analysis and a No. 3 fracture, made when the furnace was coming up out of a scrape." I asked to be shown a specimen of this iron, and being struck by its absolute similarity of appearance to the "wild" gray iron made on a coke furnace under similar conditions, I was at once convinced that it contained oxygen and that the notion of its superiority to ordinary iron was an erroneous one; and I condemned it without qualification.

I was strongly of the opinion then that the more easily reducible the ore the better would be the iron, because the more thoroughly deoxidized it would be. This was very strongly the opinion of William Wilkins, General Superintendent, and he accordingly bought some Mesabi ore of first-class physical structure, containing about 7 per cent. of combined water; in other words, a fine limonite; and we made a test with this ore, substituting it for our regular Gogebic hematite, a little at a time.

We had by that time instituted the making of test bars from every cast, and could observe directly the variations in the strength pro-

duced by different conditions. We found that the strength of the iron fell steadily as the percentage of Mesabi increased, so that it dropped from 2,800 or 2,900 lb. on a 1.25-in. test bar to about 2,400 lb. when we had on from 75 to 80 per cent. of the fine limonite. We then took off the Mesabi, got the furnace back on to an all soft-hematite burden and repeated the experiment, with absolutely identical results.

About that time my attention was called by another furnace-man to coke irons made on an all-limonite burden, which were of beautiful analysis, soft, open grained, and exceedingly weak. It then occurred to me that we had disliked the "wild" coke irons because they were generally associated with high sulphur, without regard to their physical characteristics, and, on thinking it over, I realized that they were not infrequently exceedingly strong, tough irons; and finally it seemed to me likely that by reversing my former opinion absolutely we might perhaps get a theory which would conform more closely to the facts, and that it might be the presence of a certain amount of oxygen (in what form I do not pretend to say) that produced certain characteristics as to fineness of grain, chilling qualities, strength, and toughness, which are so much desired. It is now more than two years ago that I began to work on this hypothesis with all the means at my command.

I have previously mentioned a product virtually unknown at coke furnaces, which we call spongy No. 6. This is an iron made on an exceedingly cold furnace, perfectly white, and filled with a mass of blow-holes so great as to occupy up to a half or even more of the volume of the metal. The walls of these blow-holes are as white as silver. The cause evidently is the descent into the hearth of a mass of incompletely reduced iron, which, mixing with the liquid iron already present, sets up an action almost like that in an open-hearth furnace. The silicon is removed down to the barest trace, 0.03 per cent. or so, and the carbon is then attacked and oxidized from the normal 3.50 to 4.00 per cent. down nearly to 3.00 per cent. in some cases. The iron, being very cold and low in carbon, is naturally very pasty and chills without giving a chance for the escape of the CO formed by the oxidation of the carbon.

It was evident that this iron must contain a great quantity of oxygen, and that if this were really the cause of strength and the other desired qualities in iron it could be secured by making a mixture of this spongy No. 6 and a normal iron. Such a mixture was accordingly melted in the crucible, and test bars made therefrom showed results as strong or stronger than the best iron ever made

from the furnace; 1.25-in. round bars on 12-in. centers breaking up nearly to 5,000 lb. as against an average of about 3,000 lb. for normal irons.

Attempts to Produce Special Iron at Will

It then seemed that we had only to introduce oxygen into the hearth of the furnace to make an iron as good as the best. The easiest way to do this seemed to be to cause irregular working of the furnace and this we proceeded to try out by purposely distorting the filling for several hours. We undoubtedly upset the regular running of the furnace, but the iron was no better, probably somewhat worse than before. We then put a medicine valve on the bustle pipe, filled it with mill scale and blew it into the hearth just before cast time, with little or no result, and such as there was, a detriment to the quality. This was repeated several times.

We then began ladle experiments, heating iron ore and other kinds of oxides in the bottom of a ladle before cast time and running iron in on them at cast, stirring up the mixture and pouring test bars and sample pigs. By this means we did close up the grain of the iron and induced a paper chill (that is, a line of white no thicker than a piece of paper around the edge of the pig), which was not present in the regular cast, but when it came to improvement in strength and toughness there was little or none to be observed.

During the early part of this investigation descriptions of the Ledebur apparatus for the determination of oxygen in iron and steel as modified by the chemists of the American Rolling Mill Co. were published, and we bought a set of this apparatus and began direct determination of oxygen. This determination in the presence of 3 or 4 per cent. of carbon is a very different matter from its determination in steel, and it took some time to learn all the precautions necessary to secure low blanks and concordant results.

This end, however, was attained in time. We first made a determination of oxygen in some of the spongy No. 6 and found it to contain as high as 0.13 per cent. We then took cold-blast irons and special cylinder irons and found these to range as high as 0.07 per cent. We then took normal charcoal irons and found them to contain only about 0.015 per cent.; foundry coke irons and electric-furnace irons contained none at all. We were unable to secure samples of "wild" coke basic iron such as described above, but I have no doubt that analysis would show the presence of oxygen in this.

Method of Making Remelts

As many of the data in this article are derived from tests of remelts, it seems desirable to state that all the remelts mentioned or tabulated in this paper are made in a "coke hole," approximately elliptical in shape so as to take two crucibles at once, and provided with two tuyeres at opposite ends of its long diameter, so that both its crucibles would be subjected to the same intensity of combustion, etc. In all cases where directly comparative results of a given treatment have been sought iron from the same cast has been used in each crucible, melted at the same time, and the two poured within a few minutes of each other; one with, the other without the desired treatment. We have endeavored to get two 2-in. square bars, two 1-in. square bars, and two 1.25-in. round bars, all cast on end, from every such heat, but not having a skilled molder on this rather difficult work, we have not always succeeded in getting a complete set of perfect bars. There is also, of course, a variation to be expected in these bars on account of the nature of the material and on account of their being industrial rather than laboratory or refined tests.

Results of Remelts

We repeated many times the experiment of remelting normal or even very poor ("spotted") irons, with spongy No. 6, and with the same result in every instance, an improvement in the strength up to 50 per cent or more.

We give in Table I. a summary of the results on the three pairs of remelts mentioned earlier, which shows their chemical composition and also the strength of the original iron and of the remelts.

It will be seen that in every case those irons which were strong as they came from the furnace retained their superiority after remelting. It will also be seen that the good iron was in each case much higher in oxygen than the corresponding poor iron. To summarize, therefore, we consider that we have proved the following facts:

1. Strong, tough irons of high chilling qualities contain oxygen in appreciable quantities. Weaker irons of similar analysis contain less or no oxygen.

2. Poor and normal irons may be improved beyond recognition by introducing oxygen into them.

Experiments on Deoxidation

As an additional proof that the actual cause of the increased strength of the strong irons was oxygen, we thought it would be well to try

TABLE I.

Original Iron.		Remelt No.	Breaking Strength.		T. C.	G. C.	C. C.	Si.	Mang.	Phos.	Sul.	N.	O.	Micrograph. No.	Fracture.
Cast No.	Kind.		1.25-In. Rd. Bars.	2-In. Square, Bars.											
A	Soft 1	.....	3,000 3,350	.....	4.00	3.22	0.78	1.90	0.62	0.139	0.017	0.006	0.027	1 & 2	Close grain for grade.
A	Soft 1	R. 291	3,150 3,200	16,000 15,900	4.05 4.05	3.40 3.05	0.65 1.00	Sample taken Sample taken		from 2-in. bar from 1-in. bar		.....	.....	5 & 6	.....
B	Spl. 1	.....	2,500 2,900	.....	4.09	3.43	0.66	1.88	0.70	0.112	0.020	0.0030	0.009	3 & 4	Very open, low chill.
B	Spl. 1	R. 292	2,600 2,700	13,000 13,900	4.08 4.08	3.30 3.29	0.78 0.79	Sample taken Sample taken		from 2-in. bar from 1-in. bar		.....	.....	7 & 8	.....
C	High 2	.....	3,400 3,500	.....	4.00	3.05	0.95	1.00	0.46	.....	.....	.....	0.038	9 & 10	Close grain, $\frac{1}{8}$-in. chill.
C	High 2	R. 297	3,800 3,700	16,000 15,000	3.96 3.96	3.00 2.96	0.96 1.00	Sample taken Sample taken		from 2-in. bar from 1-in. bar		.....	.....	13 & 14	.....
D	High 2	.....	2,400 2,900	.....	4.05	3.15	0.90	1.00	0.50	.....	.....	.....	0.009	11 & 12	Open grain, no chill.
D	High 2	R. 298	3,000 2,800	14,900 12,000	4.13 4.13	3.00 3.16	1.13 0.97	Sample taken Sample taken		from 2-in. bar from 1-in. bar		.....	.....	15 & 16	.....
E	Med. 3	.....	3,500 4,200	.....	4.10	3.50	0.60	0.70	0.60	0.128	0.035	.....	0.065	17 & 18	Very close grain, 0.25-in. chill.
E	Med. 3	R. 301	4,150	19,300 20,800	4.10	2.90	1.20	Sample taken		from 2-in. bar		.....	.....	21 & 22	.....
F	Med. 3	.....	2,500	.....	4.49	3.60	0.89	0.72	0.59	0.108	0.024	.....	0.009	19 & 20	Open grain, $\frac{1}{8}$-in. chill.
F	Med. 3	R. 302	2,750	10,900 12,900	4.49	3.20	1.39	Sample taken		from 2-in. bar		.....	.....	23 & 24	.....

the effect of deoxidizing some of these strong irons with recognized agents used for that purpose. We accordingly made various remelts of strong irons which we deoxidized in the crucible immediately before pouring with aluminum in several cases and with titanium in one or two. The results are shown in Table II. The results of the first trials are given by Nos. R 278, R 303, R 279, and R 304. They did not seem to us as conclusive as they should be and we came to the conclusion that the aluminum in particular, on account of its being so much lighter than the iron and in the form of fine shot, could not be properly mixed with the contents of the ladle to secure the desired result. We therefore made two additional remelts, R 305 and R 306, in which we used a piece of scrap aluminum, weighing about 1 oz., for each crucible holding 50 lb. of iron. This scrap aluminum was securely fastened to the end of a slender iron rod. It was thrust down to the bottom of the molten iron in the crucible and stirred around until all the aluminum was gone. The results in this case were decidedly more conclusive, as will be seen by the last two comparative lines in the table, but even in this case the iron was not as weak as normal irons. We therefore determined the oxygen in the test bars and found it to be 0.018 per cent. in R 305, and 0.021 per cent. in R 306, thus showing that in spite of this drastic treatment the oxygen was not completely removed and accounting for the comparatively high strength of the iron even after the deoxidizing treatment.

The results with titanium are confirmatory of the results with the aluminum. The strong irons deoxidized with about 1 oz. of 10 per cent. ferro-titanium per 50-lb. crucible were greatly weakened.

We believe that these results may be taken as a final and conclusive proof that oxygen in some form not known or understood by us is accountable for the abnormal strength of irons whose analysis as regards other elements is normal.

The Condition and Detailed Effects of Oxygen

As to the condition in which this oxygen exists, we know nothing. We have been unable to detect it with the microscope, and we can only assume that it is some oxide of very much lower degree than any with which we are familiar.

Its effects seem to be two-fold:

1. It makes the iron very much more sensitive to chilling influences. One of these good irons if cooled normally will be only slightly harder than a normal iron, and on account of its fine, close grain will machine beautifully, but if the surface be chilled even slightly it begins to whiten up, and with a chill of any strength

TABLE II.

Original Iron. Cast No.	Original Iron. Kind.	Remelt No.	Treated With	Breaking Strength. 1.25-in. Rd. Bars.	Breaking Strength. 2-in. Square Bars.	Breaking Strength. 1-in. Square Bars.	T. C.	G. C.	C. C.	Si.	Mang.	Phos.	Sul.	N.	O.	Micrograph No.	Fracture.
G	Low 2	.....	.....	3,450	20,800	.....	3.77	2.83	0.94	1.17	0.30	0.140	0.020	0.0040	0.048	.....	.....
G	Low 2	R. 277	.....	3,550 3,450	20,000	3,500	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	⅛-in. chill, close top and bottom, dark color, very good fracture.
G	Low 2	R. 278	Aluminum	3,500	17,800 14,800	.....	3.80	3.00	0.80	.....	.....	.....	.....	.....	0.012	.....	⅛-in. chill, close grain top and bottom, open center.
G	Low 2	.....	.....	3,450	.....	.....	3.77	2.83	0.94	1.17	0.30	0.140	0.020	0.0040	0.048	.....	No chill, center rather open grain, dark color fracture.
G	Low 2	R. 279	Titanium	4,000 3,900	23,900	.....	3.73	2.88	0.85	.....	.....	.....	.....	.....	0.018	.....	.....
E	Med. 3	.....	.....	3,500 4,200	.....	.....	4.10	3.50	0.60	0.70	0.60	0.128	0.035	.....	0.065	.....	.....
E	Med. 3	R. 303	Aluminum	3,800 3,200	21,700 21,600	2,600 2,850	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
E	Med. 3	.....	.....	3,500 4,200	.....	.....	4.10	3.50	0.60	0.70	0.60	0.128	0.035	.....	0.065	.....	.....
E	Med. 3	R. 306	Aluminum	4,000 4,000	18,800 18,800	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.021	.....	.....
C	High 2	.....	.....	3,400 3,500	.....	.....	4.00	3.05	0.95	1.00	0.46	.....	.....	.....	0.038	.....	.....
C	High 2	R. 297	.....	3,800 3,700	16,000 15,900	.....	3.96 3.96	3.00 2.96	0.96 1.00	Sample taken from 2-in. bar Sample taken from 1-in. bar			.....	.....	.....	.....	.....
C	High 2	R. 305	Aluminum	3,700	18,000 16,000	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.018	.....	.....
E	Med. 3	.....	.....	3,500 4,200	.....	.....	4.10	3.50	0.60	0.70	0.60	0.128	0.035	.....	0.065	.....	.....
E	Med. 3	R. 304	Titanium	3,800	16,200 17,900	2,650	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

will be solid white at the edge, where a normal iron of the same silicon would show little chill, or none at all.

2. The other principal effect is shown by the photographs, Figs. 1 to 24, and consists in the extremely fine subdivision of the graphite in the strong irons, contrasting conspicuously with the large, coarse aggregates of graphite in the weak irons. Comparing these photographs of strong irons with those of malleable iron cast white and graphitized by annealing, one is struck with the resemblance, and it is easy to suppose that the effect of oxygen is to raise the solidifying point of the iron relative to the temperature of evolution of the graphite so that the latter, instead of being free in the liquid mass to assume as bulky a form as it chooses, can only form against great pressure and must therefore form in the most compact particles possible, which, in turn cannot coalesce into larger masses because of the resistance of the surrounding medium. These are purely scientific points outside the field of a commercial investigation, and beyond our power to investigate.

A further condition which exists in these high-oxygen irons is that the tendency to form plates of eutectic and to form a spot in the center of the pig is entirely suppressed, no matter what the silicon of the iron may be, even with quite high carbons. We believe this to be a direct effect of the presence of oxygen, but as the total carbons are generally lower in the strong iron than they are in the weak ones, and as the tendency to form eutectic has been previously shown to depend on high carbon, we cannot absolutely prove whether its absence is the effect of the high oxygen content or of the low carbon content. Some of the strong irons, however, as will be seen from Table I., are quite high in total carbon; in fact, are as high as some irons in which distinct cleavage and corresponding weakness have been observed, so we feel practically certain that this highly beneficial suppression of eutectic formation is a third valuable consequence of high oxygen content.

The unvarying association of high cleavage due to eutectic formation with low chilling power in weak irons, and of high chilling power with a suppression of eutectic formation in strong irons, leads one to wonder if there is not some connection between these phenomena; whether, in fact, the eutectic may not be unstable and tend to break down into ferrite and graphite with consequent absence of chill, whereas, the austenitic formation previously shown to be characteristic of good irons may tend to establish a stable equilibrium in the pearlite conditions with consequent higher combined carbon.

These are merely suggestions for scientific metallurgists to consider, and discard or accept at their pleasure.

Comparison of High Oxygen Cast Iron and Malleable Cast Iron

Returning for a moment to the resemblance of these high oxygen irons to malleable, it may well be that this resemblance is more than accidental. The best malleable iron is made in the air furnace rather than in the cupola, and in the air furnace the conditions are entirely oxidizing, while in the cupola they are very largely reducing. A bath of white iron low in silicon at a low temperature in the oxidizing atmosphere of an air furnace might easily be capable of absorbing a certain amount of oxygen, which would tend to give a good strong white iron casting before malleablizing.

It is a matter of common knowledge that good charcoal iron makes a far better raw material for the production of malleable castings than coke iron. On the other hand, there has been much trouble in the malleable industry from attempts to use spotted charcoal irons. (See discussion by Bradley Stoughton of my paper, The Effect of High Carbon on Charcoal Iron.[2]) These are irons which I shall presently show are made under conditions tending not only to super-carburization but to thorough deoxidation, which, in fact, analysis proves to take place.

Considering all these circumstances, it seems at least possible that good malleable castings are most easily made from high-oxygen charcoal irons, because the oxygen has the tendency to throw the carbon into the combined condition, and, therefore, give the necessary chill to iron high enough in silica to be "live," at the same time throwing any graphite which may form in the original casting into the nodular form, the same as that into which it is forced by annealing.

The use of the air furnace does not completely replace the best grades of charcoal iron because its conditions are less favorable to the absorption of a large quantity of oxygen than the conditions in the hearth of a relatively cold blast furnace, which receives a certain amount of incompletely reduced iron sponge in the virtual absence of silicon, whose presence would prevent its absorption by the bath.

Reason for Low Oxygen Content Ordinarily

It seems desirable here to point out why it is virtually impossible to get much oxygen into an iron except when the furnace has been in a "mess," and to point out the relation between this fact and the commercial absence of irons of similarly high quality from coke furnaces under the same conditions.

[2] *Trans.*, xliv, 354 (1912).

FIG. 1.—UNETCHED. FIG. 2.—ETCHED.

Magnified 100 diameters.

GOOD IRON. Silicon, 1.90; Oxygen, 0.027 per cent.

Breaking strength, 1.25-in. bar, 3,000 lb., 3,350 lb.

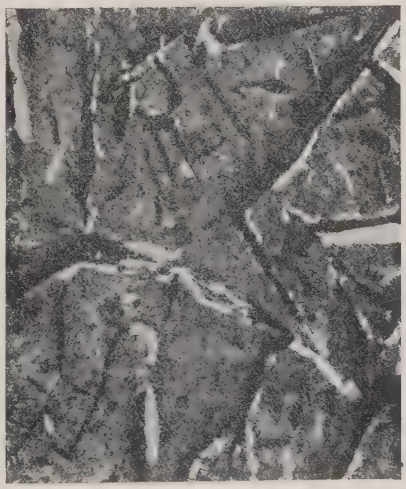


FIG. 3.—UNETCHED. FIG. 4.—ETCHED.

Magnified 100 diameters.

POOR IRON. Silicon, 1.88; Oxygen, 0.009 per cent.

Breaking strength, 1.25-in. bar, 2,500 lb., 2,900 lb.

FIG. 5.—UNETCHED. FIG. 6.—ETCHED.

Magnified 100 diameters.

REMELT. Silicon, 1.90 per cent.

Breaking strength, 1.25-in. bar, 3,150 lb., 3,200 lb.

FIG. 7.—UNETCHED. FIG. 8.—ETCHED.

Magnified 100 diameters.

REMELT. Silicon, 1.88 per cent.

Breaking strength, 1.25-in. bar, 2,600 lb., 2,700 lb.

FIG. 9.—UNETCHED. FIG. 10.—ETCHED.

Magnified 100 diameters.

GOOD IRON. Silicon, 1.05; Oxygen, 0.038 per cent.

Breaking strength, 1.25-in. bar, 3,400 lb., 3,500 lb.

FIG. 11.—UNETCHED. FIG. 12.—ETCHED.

Magnified 100 diameters.

POOR IRON. Silicon, 1.00; Oxygen, 0.009 per cent.

Breaking strength, 1.25-in. bar, 2,400 lb., 2,900 lb.

FIG. 13.—UNETCHED. FIG. 14.—ETCHED.

Magnified 100 diameters.

REMELT. Silicon, 1.05 per cent.

Breaking strength, 1.25-in. bar, 3,800 lb., 3,700 lb.

FIG. 15.—UNETCHED. FIG. 16.—ETCHED.

Magnified 100 diameters.

REMELT. Silicon, 1.00 per cent.

Breaking strength, 1.25-in. bar, 2,800 lb., 3,000 lb.

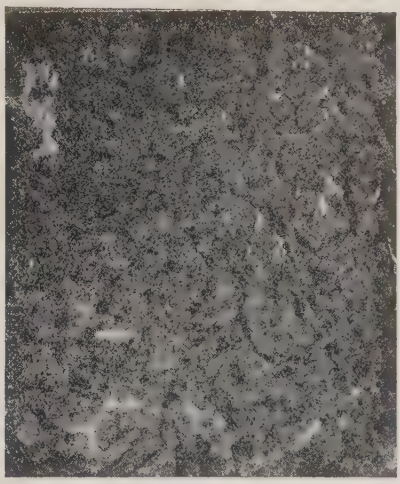


Fig. 17.—Unetched. Fig. 18.—Etched.

Magnified 100 diameters.

Good Iron. Silicon, 0.70; Oxygen, 0.065 per cent.

Breaking strength, 1.25-in. bar, 3,500 lb., 4,200 lb.

Fig. 19.—Unetched. Fig. 20.—Etched.

Magnified 100 diameters.

Poor Iron. Silicon, 0.72; Oxygen, 0.009 per cent.

Breaking strength, 1.25-in. bar, 2,500 lb.

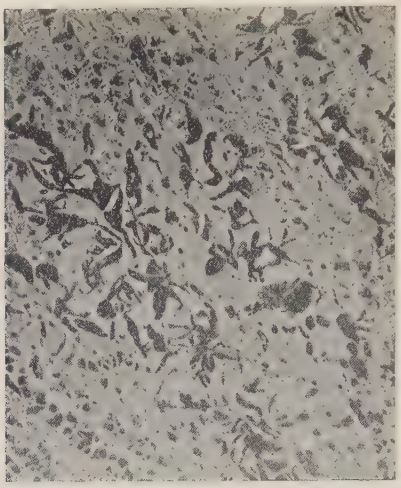


FIG 21.—UNETCHED. FIG. 22. —ETCHED.

Magnified 100 diameters.

REMELT. Silicon, 0.70 per cent.

Breaking strength, 1.25-in. bar, 4,150 lb.

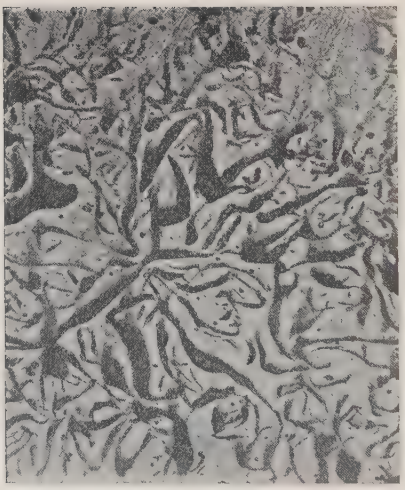


FIG. 23.—UNETCHED. FIG. 24. —ETCHED.

Magnified 100 diameters.

REMELT. Silicon, 0.72 per cent.

Breaking strength, 1.25-in. bar, 2,750 lb.

FIG. 25.—ELECTRIC FURNACE PIG IRON.
Magnified 100 diameters.

FIG. 26.—ELECTRIC FURNACE PIG IRON.
Magnified 100 diameters.

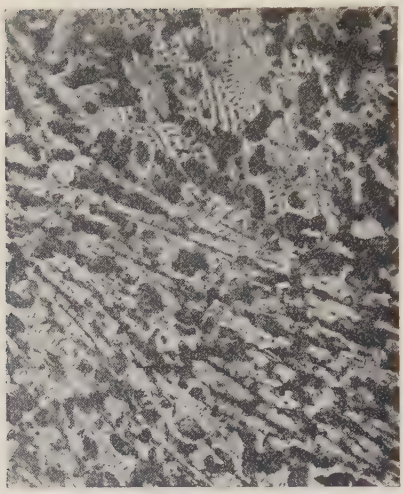


FIG. 27.—CHROME, 4 PER CENT.
Iron used as base was extra good special cylinder. Compare this with Fig. 14.

FIG. 28.—CHROMIFEROUS PIG.
Note resemblance to spotted weak iron in Fig. 27.

FIG. 29.—TYPICAL SPOTTED IRON.
POOR IRON (Spotted). Etched. Magnified 100 diameters.

The increase in the affinity of carbon for oxygen is well known to be extremely rapid with increase in temperature. The affinity of silicon for oxygen and the ease with which silicon may be oxidized by oxides of iron is similarly well known. When, therefore, we attempted to introduce oxygen into the iron by disturbing the regularity of the working of the furnace or by blowing in oxide at the tuyeres, we did introduce iron oxide into the bath; but, in the presence of an ample supply of heat and incandescent carbon, this oxide was instantly attacked by either the silicon or the carbon and reduced to metallic iron, which passed into the bath without effect. The same thing happened when we introduced heated iron oxide into a ladle full of iron. The oxide attacked the silicon in the bath and was reduced to metallic iron, with the oxidation of silicon to silica. Probably also carbon may have been attacked to some extent, but no oxygen ever became united with the iron of the bath because of the superior affinity which silicon and carbon have for it at these temperatures.

It is, however, a well-known fact that once iron has taken up oxygen it is exceedingly difficult to remove it even with powerful deoxidizing agents. The steel makers, I believe, now recognize that they get occasional casts of iron of faultless analysis from which good steel cannot be made in the Bessemer or open-hearth furnace, no matter how much deoxidizer may be used, but these irons may be completely deoxidized and made into first-class steel in the electric furnace on account of its power to maintain a reducing atmosphere and an exceedingly high temperature for an indefinite period.

Conditions for High Oxygen Content

It seems, therefore, that when the temperature of the hearth of the furnace has become lowered by the descent into it of a mass of relatively cold and imperfectly reduced material, this low temperature and the practically complete absence of silicon permit a certain amount of oxidized material to be dissolved in the bath in spite of the presence of more than 3 per cent. of carbon. It would seem as though this material were an imperfectly reduced iron sponge, which, either from a deficiency of heat or a deficiency of carbon in the hearth, had never given up its last traces of oxygen, but passed into the bath in this condition. This material, having once become incorporated with the iron, is not deoxidized by any ordinary subsequent reaction.

If, now, the furnace be cast while this condition prevails we shall get spongy No. 6, but if the iron be held in the hearth until additional fuel comes down and restores the regular operation of the

furnace, the normal iron so made will descend into the bath of spongy No. 6, will liven it up and impart silicon to it, and will, perhaps, reduce, but will not completely eliminate, the oxygen it contains. If the furnace be cast after these conditions have prevailed for a sufficient period, we shall get a cast of special cylinder iron.

When we remelt a normal iron with the spongy No. 6 in a crucible, we do no more than repeat on a small scale what the furnace does on a large scale when the conditions are right.

Coke-Furnace Conditions as Regards Oxygen

In a coke furnace the conditions are entirely altered, because, as is universally understood, such a reduction in temperature with that fuel is almost inevitably accompanied by a rapid increase in the absorption of sulphur by the iron, and this element seems to introduce a totally different set of conditions.

This is probably the result of two facts:

1. That the hearth temperatures in coke furnaces are necessarily much higher than those in charcoal furnaces on account of the greater infusibility of their slags, and even after such reductions are far above that at which spongy No. 6 is found in charcoal furnaces.

2. Because it seems probable that the sulphur combines with the oxygen and that the oxysulphide has a vastly different effect on the quality of the product from that of oxygen in the absence of sulphur.

I do not wish to be understood as claiming that we have proved that oxygen will unite with sulphur when the latter is present to any considerable extent, and so form an oxysulphide; for I have no direct proof to offer that this takes place. At the same time, the marked difference between the sulphur by the evolution method and that by the gravimetric method in "wild" coke irons, above described, seems to me a strong indication in that direction. Moreover, the wild coke irons seem to follow a different line of development from the wild charcoal irons; for instance, spongy No. 6, which is not infrequently made at charcoal furnaces, is almost unknown at coke furnaces. Those who have made medium sulphur white irons in a coke furnace ("basic") know that this iron when broken shows oxide colors on all the cleavage faces. These are so pronounced and brilliant (ranging from red to purple) as to eliminate the white appearance of such iron completely in some cases, and a very wild coke iron cast into a chill sample pig and cooled off quickly with water will break up into little cubes like dice, of which the faces are every color from purple to black, but practically never white.

These phenomena are practically never present in charcoal irons. The temper colors are rarely seen. The wildest of all the iron made, spongy No. 6, shows little or no tendency to break up into cubes, and in general there is but little resemblance between the two varieties of wild iron, whose only important difference, so far as known to me, is the considerable quantity of sulphur in the one and the small quantity in the other.

In the course of close observation of coke iron extending over a number of years, I came to the conclusion that the effect of oxygen was to augment the effects of sulphur so that a wild iron with high sulphur was more objectionable; that is, weaker and more prone to break into small pieces by contraction cracks when hard, than an iron with the same sulphur but not made on a wild-working furnace. This view is not capable of proof by absolute evidence, for unfortunately I cannot offer any such as that with which I have aimed to support my more positive conclusions. At the same time, a conclusion of this kind based on years of careful observation has to my mind a certain value, and I am submitting it here for what it may be worth.

The Heat Balances of Different Kinds of Furnaces

It seems desirable to point out here certain factors concerning the heat balance of the blast furnace with different fuels and different slags, as well as different blast temperatures, because they affect this problem vitally.

By the heat balance I do not mean merely a statement of the total number of thermal units developed and introduced by the blast minus those carried out by the top gases, but a balance taking account of the quantities both of high-temperature and of low-temperature heat developed in any furnace under given conditions.

The theory that the blast furnace has a critical temperature, which is in effect the free-running temperature of the slag, and that the heat necessary for the hearth reactions can only be that available in cooling the gases formed at the tuyeres from their theoretical combustion temperature down to this critical temperature, seems to have been accepted widely enough to justify its use for making a more instructive heat balance.

We have been able, ever since Bell's researches, or before, to calculate the total heat imparted to the furnace. By the use of this critical temperature theory we are able to figure the amount available for the hearth under any given conditions when the free-running temperature of the slag is known. It is obvious that there must be some more or less distinct relationship between these two

quantities of heat. A certain amount is necessary for the reduction of the ore and for bringing the iron and slag-forming materials up to the critical temperature. A certain further amount is necessary for actually melting them and for the completion of the reduction process in the broadest sense.

It is obvious that the ratio which these two quantities must bear to one another can only vary to a considerable extent through a great variation in the character or quantity of the slag, the nature of the ore, or some other fundamental variation of furnace conditions.

I formerly called the high-temperature heat utilized in the hearth and bosh the "available" heat, but for the purposes of this discussion I now propose to substitute for this the term "hearth" heat, because I shall show that some of this available heat passes up out of the hearth and so becomes "available" in the shaft of the furnace, thus leading to confusion. The heat applied for purposes of reduction and preparation in the shaft of the furnace I have called "shaft" heat. I recognize the clumsiness of these terms and will accept with gratitude any better ones which may be offered.

The hearth heat is found by adding to the heat of combustion of the fuel burned in the hearth that brought in by the blast, and deducting that required for the dissociation of the moisture in the blast and for raising the temperature of all the gases of combustion to the critical temperature.

The shaft heat is the last-mentioned sensible heat in the gases of combustion, plus the heat developed in the shaft by the further oxidation of the gases from the fuel by oxygen from the ore, less the sensible heat carried out by the top gases.

Pyrometric observations carried on for most of the past three years have indicated the free-running temperature of charcoal slag to be about 2,300° F., as against about 2,700° F. for coke slags in ordinary practice.

Comparing the results of the dry-blast coke run, published in James Gayley's paper on that subject some years ago, with our own practice, I found that the coke furnace at Isabella was developing about 1,000 thermal units of hearth heat per pound of iron, while in our practice at that time we were developing between 1,100 and 1,200, although our slag volume was only about one-half of that at Isabella.

This surprising result, after careful consideration, could mean only one thing, that we were developing in the hearth more heat than was required for the operations of the hearth; that part of the excess was expended in superheating the iron and slag above the temperature

necessary, the balance passing up into the shaft of the furnace and being utilized there, although at a lower temperature.

An exhaustive calculation of heat balances in coke and charcoal furnaces in the sense above described, indicated very clearly that the limiting quantity of heat in the Lake Superior charcoal furnace was not in the hearth, as in coke furnaces, but in the shaft. This is borne out by the fact that a change in blast temperature of 300° or 400° on such a charcoal furnace will only change the iron a couple of grades, whereas, in coke practice it would mean almost the difference between regular operation and "chilling." This is obviously for the reason that only a certain definite supply of hearth heat is necessary and after the heat of the blast has completed this supply any additional heat in the blast serves only as additional shaft heat and the saving can only be reckoned in percentage of the total shaft heat. The saving is therefore only about one-fourth as great as that made by an equal quantity utilized as hearth heat.

In the case of the hot-blast charcoal furnace above described, the heat developed in the hearth is actually capable of taking care of much more ore, but the gases in the shaft are not able to reduce any more because they are cooled far below the temperature of possible reaction when they leave the top of the furnace, not infrequently to 200° F. or below. For convenient reference let us call this Case I.

The consequences of this condition are far-reaching and are difficult to realize by those accustomed only to coke practice on lean or refractory ores, where every increase in the hearth heat corresponds to an exactly proportionate reduction in fuel consumption. In those furnaces there is always a great excess, either actual or potential, of shaft heat; that is to say, if the top gases are not themselves so hot as to be capable of much further action on the ore, they are so rich in carbon that they could be oxidized to a far greater extent with the oxygen from additional ore, if only the heat developed in the hearth were able to take care of the ore so reduced. (Case II.)

These two types of furnaces, hot-blast charcoal, and coke with lean or refractory ores, represent two opposite types as far as their heat balances are concerned. The former has a relative deficiency of heat in the shaft, the latter in the hearth.

There are two other groups of furnaces, the first of which belongs to one of these types, the second to the other. These are the coke furnace using rich and easily reducible ores, and the cold-blast charcoal furnace.

In the latter the hearth heat developed per pound of fuel is only

about four-tenths of that developed in the hot-blast charcoal furnace, so that while the amount of fuel used in these furnaces is much larger than that used in the hot-blast furnaces, the hearth heat developed per pound of iron is so much less than in that case as to make the heat balance like that of the coke furnace on lean ores; that is, to make a marked deficiency in the hearth as compared with that developed in the shaft. (Case III.)

The first group above mentioned, coke furnaces on rich and reducible ores, embraces those furnaces making the greater portion of the pig iron output of the country. They are in general well equipped with stoves and it is notorious that it is often impossible for them to use all the heat that they can get, because the operation of the furnace is deranged by anything more than very moderate blast temperatures. These furnaces, on the other hand, work with very cold tops; the gases commonly passing off at a temperature well below that at which any reaction on the ore can occur, no matter how rich the gas. (Case IV.)

These two facts may be taken as a proof, in a practical way, of the correctness of the results of an accurate calculation of the heat balance, which shows that in these furnaces there is generally an excess of heat in the hearth.

The Effect of Different Types of Heat Balance on Quality of Iron

Since the quantities of carbon and of oxygen present exercise so large an influence on the quality of the iron, it is worth while to examine the conditions in these four cases with a view to their effect on these two elements.

We know but little about the laws which control the absorption of carbon by the iron, but two facts are established:

1. The greater the quantity of carbon the greater the opportunity for saturation at the given temperature and the more rapidly saturation will occur.

2. The higher the temperature the greater the degree of saturation possible.

We also know, in a general way, some of the laws of deoxidation, of which two may be roughly stated as follows:

1. The greater the exposure to carbon the more complete the deoxidation at a given temperature.

2. The higher the temperature the more complete the deoxidation with a given exposure to carbon.

Case I. A poorly working hot-blast charcoal furnace in which poor distribution, or other irregularity with consequent imperfect reduction, is sought to be overcome by high blast temperatures and high fuel, furnishes the two conditions most favorable for complete deoxidation and high carburization, leading to a hyper-eutectic iron, with its consequent tendency to form eutectic plates and lose its strength thereby. It is under these conditions that we obtain the "spotted" irons of well-deserved bad reputation.

The obvious corollary to this is that when we have a regularly working furnace with a minimum of sudden changes and calls for extra heat to keep it out of trouble, we shall make a better grade of iron. In other words, good furnace work is the first requisite to high quality. In addition to this we can cut down our blast temperature, which, as is well understood, reduces rapidly the hearth heat developed per pound of fuel, until we reach a point where both sides of the thermal equation balance, and at this point we shall have the lowest hearth temperature and the lowest degree of superheating of the iron and cinder, combined with a comparatively small volume of fuel in the hearth.

Further, the inferior quality obtained by our experiments with the easily reducible fine limonite from the Mesabi tends to show that a quantity of hearth heat which may not be too great to give good quality with one kind of ore is excessive for a more reducible variety of ore.

Case II. The coke furnace, on lean or refractory ores, suffers chronically with a deficiency of heat in the hearth. The hearth temperature and that of the iron are closely controlled by the free-running temperature of the slag, and there is little or no superheating.

(Superheating is used in this discussion to mean heating above the free-running temperature of the slag.)

On the other hand, we have a higher temperature and a much greater quantity of fuel in the hearth than in the hot-blast charcoal furnace.

These two factors tending to more perfect deoxidation offset the advantage (from the quality point of view) of having a deficiency of heat in the hearth and these irons are accordingly not so good as charcoal irons; but that they may occasionally contain oxygen is, in my judgment, abundantly indicated by the evidence I have cited, and the fact of their high reputation among coke irons in the foundry trade for the desirable qualities of strength, toughness, and density of grain is too well established to be gainsaid.

This case represents, in a general way, Eastern and Southern coke irons, and the premium which these command in the open market is a sufficient proof of their superiority to the ordinary coke irons made from the Lake Superior ores.

Case III. The cold-blast charcoal furnace uses a considerable quantity of fuel, but so little hearth heat is developed from it that a chronic deficiency exists. For this reason these furnaces run continually on the brink of disaster. Their slag is heated barely to the point where it can be removed from the furnace with the assistance of hooks, hence their hearth temperature is so low that deoxidation, even in the presence of the considerable volume of fuel, cannot be complete, consequently the irons of this variety that we have analyzed always contain considerable percentages of oxygen and are well known to have the highest chilling qualities and strength obtainable with any iron. At the same time, these irons have an ample opportunity for absorbing carbon, and as a matter of fact, those we have analyzed run quite high in that element, frequently around 4.00 per cent. But the detrimental effects of this are eliminated by the presence of oxygen. The latter, as above mentioned, seems to prevent the formation of the eutectic, which causes weakness in high-carbon irons when low in oxygen.

Case IV. Coke furnaces on rich and easily reducible ores have frequently a surplus of hearth heat (as indicated by the inability of the furnace to use a higher blast temperature), a volume of fuel in the hearth not much less than that of the lean-ore coke furnaces, and a degree of superheating of their product which is very striking to the eye of one accustomed only to the corresponding temperatures in Case II.

These are obviously the conditions which lead to the most complete deoxidation and the highest degree of carburization, therefore to poorest quality.

The "rottenness" of castings made from ordinary Bessemer iron in the Lake ore district, especially since the introduction of large percentages of highly reducible Mesabi ore into their mixtures, is accepted as a matter of lamentable but unavoidable fact by those who, like steel-works engineers, have to use such castings exclusively.

Reviewing all four of these cases, we see that they fall naturally into two quite distinct classes:

1. Those furnaces which generate more hearth heat than they can utilize in the hearth and whose controlling limitation to fuel economy is the amount of heat they can develop in the shaft. These conditions

lead to superheating the iron and furnish the best conditions for complete deoxidation and supercarburization. To this class (which we will call for convenience Class I.) belong the hot-blast charcoal furnace and the coke furnace on rich and easily reducible ores.

2. Those furnaces which have a relative deficiency of heat in the hearth and which generate, either actually or potentially, more heat in the shaft than they need. These furnaces do not in general superheat their iron and have a tendency to work cold in the bottom, the condition most favorable to preventing complete deoxidation and supercarburization. To this (Class II.) belong the coke furnace on lean or refractory ores and the cold-blast charcoal furnace.

The iron of Class II. made with either kind of fuel is superior in reputation and test to the iron of Class I. made with the same fuel, but charcoal iron has two vast advantages over the coke iron in both classes:

1. Because of the absence of any considerable amount of sulphur, which can mask or destroy the beneficial action of the oxygen.

2. Because the critical temperature is so much lower than that of a coke furnace under the best conditions, because the charcoal furnace does not suffer under the necessity of using the calcareous and relatively infusible slag necessary for desulphurization in coke practice. This gives charcoal iron the possibility of securing always a lower degree of carburization and a higher oxygen content than is possible with the coke furnace, but this possibility has in the past often been wasted because quality has been largely disregarded in Lake Superior charcoal iron until within the past three years.

Moreover, it is only as the result of the present investigation along two lines simultaneously, the complete chemical analysis of the different qualities of iron, particularly for carbon and oxygen, and the detailed investigation of the heat balances of different conditions of furnace operation, showing how these affect deoxidation and carburization, that it has been possible to reach these conclusions, which seem to me to reconcile practice and theory to a greater extent than has been possible hitherto. The possibilities which the discovery of these facts offers for improvement in the quality of charcoal iron will be seen to be enormous, and those who have used these irons for many years seem willing to admit that we have already effected much improvement.

In accordance with the two principles of not using an excessive blast temperature and of using an admixture of an ore even less reducible than our normal supply, we have been able to make irons of the chilling grades closely approaching "special cylinder" in quality

in the regular operation of the furnace and without having to await "a mess" to produce it. These irons we believe will equal the best that can be made at the Southern and Eastern furnaces, except those using cold blast.

The electric pig iron furnace furnishes an extreme case of a furnace running with an excess of heat in the hearth. Authoritative figures for the furnace at Trollhätta give its consumption of charcoal as about 800 lb. per ton, while the electric current consumed is equivalent to about 2,500 thermal units per pound of iron against 1,000 for a coke furnace making a much larger slag volume.

The characteristic of the electric furnace is that the temperature of application of its heat may just as easily be high as low. Nearly all the heat in these furnaces is generated in the hearth and the excess not needed there is carried up through the charge column by the gas, and only a slight amount is produced by additional oxidation by oxygen from the ore.

These conditions lead to enormous superheating of the iron and to consequent perfect reduction, in spite of the relatively small quantity of fuel present. The result of this complete deoxidation and of the excessive temperature at which these irons are made is a very high degree of carburization, a practically complete absence of combined carbon, excessive quantities of graphite and the latter extending in an almost continuous network through the iron.

This, in conjunction with its low combined carbon, makes it exceedingly weak. I am informed on the authority of S. T. Wellman that this iron as it comes from the furnace has a tensile strength of only about 18,000 lb.

In Figs. 25 and 26 are shown photomicrographs of electric furnace pig iron, etched and unetched, which are reproduced here from *Chemical and Metallurgical Engineering* through the courtesy of H. M. Boylston, of Boston.

These photomicrographs show the same characteristics as the poor charcoal irons, but in a more exaggerated degree.

This furnishes a complete confirmation of what has been said above concerning the furnace conditions under which good and bad irons are made.

High vs. Low Carbon

Just here it seems worth while to say a word concerning the relative merits of high carbon and low carbon, about which there has been no little dispute, and concerning which but little appears to be known further than that some customers insist on having high carbon

irrespective of anything else, whereas, others want the carbon as low as they can get it.

The difference appears to lie in the use to which the iron is put. If it is to be chilled, then the more carbon present the greater the percentage of cementite in the chilled portion and the harder the latter will be, other things being equal. The absence of a sufficient amount of carbon from chilled castings made with a large percentage of steel scrap may well be one of the causes for the poor reputation of such chilled castings as compared with those made with low-silicon charcoal irons.

If, on the other hand, a special iron be desired for strength and closeness of grain (but not chill), as in steam or gas engine cylinders, the higher the carbon the worse it makes matters, because the strongest mixture must be one containing approximately 1 per cent. of combined carbon, and the greater excess there is over this amount the greater will be the amount of graphite, with its tendency to gather into larger aggregates as its quantity increases, so diminishing both the strength and the tightness of the casting.

The Effect of Other Elements

During the course of our investigation the effect of various other elements besides oxygen was investigated also, and the results are believed to be worth recording here, though of less importance than those on oxygen.

Nitrogen

Analyses for this element by the customary method met with a difficulty which greatly delayed the investigation along this line. The hydrocarbons formed by the solution of the sample in hydrochloric acid passed over into the Nessler solution and gave it a milky, yellowish appearance, making comparison of the color with that of a similar tube containing pure ammonia impossible. This difficulty was finally overcome by adding potassium permanganate to the acid solution and boiling it for a considerable while, then overneutralizing with caustic soda and boiling further to drive over the ammonia. We found various quantities of ammonia, which, calculated back to nitrogen, amounted to about 0.01 per cent. as a maximum, with quite wide variations. Generally, the highest quantities were present in the irons made on the coldest furnace, but we seemed unable to get any definite relation between nitrogen and quality.

We then made twin remelts of various irons, as above described, using the same irons in both crucibles, but adding a considerable

quantity of potassium cyanide to one of the crucibles a short time before pouring. The effect on the chill block cast from these remelts was distinctly marked, increasing the chill and the fineness of the grain and giving a purplish appearance to the zone between the chill and the gray; but the strength of the bars was not materially affected, even by greater quantities of nitrogen than we had ever found in the iron from the furnace, and we reached the conclusion that the effect of nitrogen was not of much industrial importance in cast iron.

Cyanogen

It occurred to us that this compound might exist in the iron, as its presence in the hearth and bosh of the blast furnace has often been proved. We made analyses by several different methods which should have shown any cyanide if present, but with a purely negative result.

Phosphorus

Two twin remelts were made, as above described, using as a base in one case a spotted iron, in the other a special cylinder iron. To one crucible in each remelt was added enough ferro-phosphorus to raise the percentage of this element to about 0.6. The chill was markedly increased in the special cylinder iron, which is more notable in view of the normal high chill of these irons.

The iron was quite low in silicon, so that the chill block was chilled clear through in both cases, but the quality of the phosphorus chill was far better to the eye than that of the straight remelt. There was a marked reduction of the size of the crystals and a tendency to pass from the flat and crystalline to the conchoidal and smooth fracture. The strength was increased about 20 per cent. in the poor iron and about 5 per cent. in the good iron. The results are shown in Table III.

We find here again that remarkable relationship between high chill and strength upon which I have so frequently commented.

The obvious explanation is that the iron with a higher combined carbon is the stronger within wide limits, but that this does not tell the whole story in all cases is conclusively proved by the results of Table I. above cited.

In the case of phosphorus the ultimate reason for the effect is as little known to us as in the case of oxygen, but we may make a suggestion. The effect of phosphorus in increasing fluidity is so well known as to need no comment here. This presumably means lowering the freezing point of the last-freezing portion; that is, the eutec-

Table III.

Original Iron. Cast No.	Original Iron. Kind.	Remelt No.	Treated With	Breaking Strength. 1.25-in. Rd. Bars.	Breaking Strength. 2-in. Square Bars.	Breaking Strength. 1-in. Square Bars.	T. C.	G. C.	C. C.	Si.	Manganese.	Phosphorus.	Sulphur.	N.	O.	Micrograph No.	Fracture.
G	Low 2	.....	.....	3,450	.....	.....	3.77	2.83	0.94	1.17	0.30	0.140	0.020	0.0040	0.048	.....	.....
G	Low 2	R. 277	Straight remelt.....	3,550	20,800 20,000	3,500	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1/8-in. chill, close top and bottom, dark color, very good fracture.
G	Low 2	.....	Ferro-phos. 5 per cent.....	.....	21,800 15,900	3,500 3,300	3.80	2.81	0.99	.....	.....	0.680	.....	.....	.....	.....	Almost white with mottled center; not a cleavage fracture.
G	Low 4	R. 263	Straight remelt.....	.....	10,000 9,700	.....	4.28	.....	.....	1.30	0.28	0.146	0.035	.....	.....	.....	1.5-in. chill ; high-cleavage iron.
G	Low 4	R. 262	Ferro-phos. 5 per cent.....	.....	12,000 11,900	.....	4.07	.....	.....	1.25	0.29	0.618	0.031	.....	.....	.....	1.5-inch chill ; slightly shown cleavage much less marked.

Table IV.

Original Iron. Cast No.	Original Iron. Kind.	Remelt No.	Treated With	Breaking Strength. 1.25-in. Rd. Bars.	Breaking Strength. 2-in. Square Bars.	Breaking Strength. 1-in. Square Bars.	T. C.	G. C.	C. C.	Si.	Manganese.	Phosphorus.	Sulphur.	N.	O.	Micrograph No.	Fracture.
G	Low 2	.....	.....	3,450	.....	.....	3.77	2.83	0.94	1.17	0.30	0.140	0.020	0.0040	0.048	.....	.....
G	Low 2	R. 277	.....	3,550	20,800 20,000	3,500	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1/8-in. chill, close top and bottom, dark color, very good fracture.
G	Low 2	R. 281	2.28 per cent. Chr.	.....	19,200 21,100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1/4-in. chill, slight tendency to cleavage
G	Low 2	.....	.....	3,450	.....	.....	3.77	2.83	0.94	1.17	0.30	0.140	0.020	0.0040	0.048	.....	.....
G	.....	.....	4.00 per cent. Chr.	.....	21,900 18,800	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

tic, in case the iron has a tendency to form a eutectic. Lowering the freezing point of this constituent would give a better opportunity for the first-freezing austenitic constituent to solidify and knit itself together so as to resist penetration by the plates of eutectic and to force the latter into small isolated pools of approximately equal dimensions in all directions; instead of permitting its formation into plates of great length and breadth, but little thickness.

It is easy to detect the presence of the phosphide eutectic in irons containing 0.50 per cent. or more of this element, and the structure of this eutectic, judging by its appearance, seems to be very different from that of the carbon eutectic. It is, in fact, very much more like pearlite. This also is probably an element of strength to the mass as a whole.

These hypotheses are not intended to be taken as statements of facts, but only as suggestions, which scientific investigators may attack at will and confirm or disprove according to the evidence.

I have previously mentioned the fact, first brought to my attention by R. W. Durrett, that phosphorus has a strong influence in reducing the total carbon of the iron. This may come from an effect similar to that of silicon in simply crowding out any excess, or it may be due to increased fluidity, which permits the iron to melt at a lower temperature and escape from the superheating and carbonization conditions above the tuyeres more quickly.

Chromium

Two twin remelts were made with chromium, using special cylinder iron, the same as No. 277, as a base. In one case there was added to one of the crucibles enough chromium to raise the percentage of this element to 2.18 per cent.; in the other to 4.00 per cent. The results are given in Table IV. It will be seen that the strength is but slightly increased over that of the same iron untreated, while the combined carbon is very materially increased, as shown by the chill and by the fracture of the bars. The chill, of course, was increased also, but not to the extent we had expected with these percentages. An interesting fact is that the fractures of the test bars showed under the microscope the eutectic structure so fully illustrated last year in connection with spotted and high cleavage iron, as shown by Fig. 27. This may be compared with Fig. 14, although the original iron of the chromium remelt was considerably better iron than that of Fig. 14; incidentally, the oxygen was also much higher.

We secured for examination a pig of an iron now being recommended

for its chill and strengthening qualities on account of its chromium content, and were struck at first glance by the presence of a white spot in the center of the pig. We accordingly made a photograph of the iron from an edge of this pig, which is shown in Fig. 28. Beside it, Fig. 29, is reproduced Fig. 5 from my paper of last year. The conclusion seems irresistible that while chromium throws carbon into the combined condition and therefore increases the strength within certain limits, it also causes the formation of a eutectic similar to, if not identical with, that which forms in high-carbon irons. Reasoning by analogy, as well as by results from the test bars, with their relatively insignificant increase in strength for a very considerable increase in combined carbon, we seem justified in believing that while chill and some strength may be secured by the addition of chromium, it will never be possible to obtain simultaneously by this means all the qualities for which the better grades of charcoal iron are in demand. These qualities we now attribute to their high content of oxygen; the high temperature and strong deoxidizing conditions necessary for the reduction of chromium are, to say the least, unfavorable to the simultaneous presence of this element and of oxygen.

Manganese

By a fortunate chance we have had to make for commercial reasons two or three runs on an iron of much higher manganese than common; in one case going up to standard spiegel. On each occasion when the manganese has risen somewhat beyond 1 per cent. we have been able to perceive at once in the chill blocks and test bars made from the cast an increase in chill and an increase in strength, but the chill is of the high-cleavage variety and is accompanied by high carbon, as is well understood in the case of spiegel. An examination of the shining plates of spiegeleisen ("looking-glass iron") will show those interested the high-cleavage structure to which I have so often alluded; in fact, on the broken ends of some pigs of this material plates scarcely thicker than a sheet of writing paper and almost as large as one's thumb nail may be seen sticking out from the body of the iron. These we take to be the eutectic of iron, manganese, and carbon, exactly similar to those already described for high carbon and for chromiferous irons.

Reason for Poor Quality of Manganese and Chrome Chills.

It seems likely that this characteristic structure may account for the poor wear given in service by chilled objects made with ordinary irons and ferro-manganese or ferro-chrome additions.

In car wheels it is well understood that the strength and chill prescribed by the Master Car Builders can be obtained by this means, but it is also a matter of sorrowful knowledge to the railroads that these wheels will not give the wear in service obtainable from wheels made of good charcoal iron, and that shelling out is their great weakness. Their structure is fundamentally one composed of these flat plates massed together, but not interlaced, because they seldom penetrate one another. It is natural that under the combined stresses of temperature due to braking and the rolling stress due to moving contact with the rail, separation should occur along these cleavage planes at points some little distance removed from one another, and that these cleavages should eventually come to form the boundaries of a section of tread, which becoming more and more loosened finally becomes detached altogether, causing a shell-out.

The same thing applies to the surface of chilled rolls but not to the same extent, because the intensity of stress per unit of area is less severe.

Conclusions

1. Irons which contain oxygen to a considerable extent, in a form as yet unknown, are stronger and tougher and have better chilling qualities than irons containing less of this element, but of the same analysis in other respects.

2. Irons may contain a sufficient quantity of carbon to make them rotten and worthless through the predominance of the eutectic structure when deficient in oxygen, and yet, when they contain a considerable quantity of oxygen, be entirely free from this structure and of excellent quality.

3. Furnace conditions control whether the carbon shall be below or above the eutectic ratio and the presence of much or little oxygen, the two dominant factors in the fundamental quality of the iron; that iron, broadly, being the best which is made at the lowest temperature.

4. Charcoal irons can be made at temperatures several hundred degrees lower than are practicable with coke and have a great possible advantage in that respect. This is largely wasted without good furnace work in the broadest sense.

5. Phosphorus up to 0.50 per cent. or more exercises a beneficial influence on the strength of the iron and the depth and character of the chill. Phosphorus also has a tendency to reduce total carbon.

6. Aluminum and titanium tend to reduce the strength and chilling power of the iron by removing its oxygen, but titanium may have an influence independent of this on account of its removing nitrogen also.

7. Chromium and manganese have an almost identical effect in raising the total carbon and throwing it into the combined condition. The latter increases the chill and the strength, but the combined carbon so produced is in the form of eutectic plates, which deprive the iron of much of the strength which it would have with the same combined carbon in the absence of these plates.

8. Chill produced by these elements cannot be expected to have the same wearing qualities as that produced by elements which do not further the formation of flat plates of eutectic.

This paper would not be complete without some acknowledgment of the valuable collaboration of L. Selmi, Chief Chemist of the Ashland plant. Mr. Selmi made all the sections and photomicrographs used in this paper, made most of the analyses, and gave the work his interest and co-operation in every way.

Discussion

Richard Moldenke, Watchung, N. J.—I am very deeply interested in Mr. Johnson's paper, as I have been working on the same subject for over 20 years. The paper is a difficult one to discuss, as there is so much in it that agrees with our experience, and again certain things that are diametrically opposed to what good foundry practice has indicated as being the case.

I must thank Mr. Johnson for proving so clearly the actual presence of oxygen in cast iron, as I have been claiming this for years as the cause of most of the foundryman's troubles on the iron end. It looks somewhat dubious to the chemist to claim the presence of even small percentages of oxygen where the carbon runs up to, say, 4 per cent., and hence steel metallurgists cannot see it. Now, however, that the analyses given have corroborated my own results along defective malleable practice, the presence of oxygen in high-carbon irons will be more readily believed. My own analyses and investigations make me believe the oxygen to be present in Fe_3O_4 dissolved or in some way present in the bulk of the metal. Mr. Johnson describes flakes of iron oxide coming out of the molten metal, to the top, and that explains a lot to the foundryman who deals only with the remelted pig iron.

I made some 14,000 analyses of low-silicon, high-carbon cast irons, pouring the samples into shot over a pail of water, picking out good pieces, about pea size, and perfectly bright and clean. When crushed and dissolved in the mixed acids for the silicon determination, no graphite being present, I noticed many years ago that whenever we had a bad run of metal—even with the proper chemical composition—there would appear on the bottom of the evaporating dish small black particles of what turned out to be the above-mentioned oxide of iron. Sometimes these periods

of poor metal ran for months, and the study of the problem finally developed the fact that a radical reform in the melting of the malleable mixtures was required, so that an oxidation of the metal could not become too serious. Short heats, careful piling of the charges to insure this, and arranging the bottom to prevent feather-edged baths, accomplished the result, gave far stronger and more perfect castings than were ever obtained before; and hence oxygen was kept out by every means possible.

Mr. Johnson, however, is perfectly right in stating that oxygen gives a harder iron. The chill becomes deeper. It makes the metal set quicker, and hence from the standpoint of hard castings the presence of high percentages—comparatively speaking—of oxygen in the iron is desirable. Unfortunately, however, the presence of oxygen allows the opportunity for a reaction with carbon, forming gas, which means pinholes and gas pockets in the castings. These will be found just under the cope, or upper part of the castings, and, when in castings that can be machined, present a sorry spectacle after an eighth of an inch of the skin is removed. The raising of the freezing point of the metal, making what is called a "dead," or lifeless, iron, means that when poured the metal in the gates sets so fast that before the casting has time to be properly fed full of metal the gates are frozen, and the casting is full of shrinkage spots. Occasionally in very highly oxidized iron the metal sets so fast with the slight lowering of temperature that molds are short-poured, and ladles are skulled up badly. These are the objections to what looks like an advantage at first glance.

I further think that Mr. Johnson's tables of analyses have a weakness in the carbon determinations that leads to defective conclusions. The sample for combined carbon has been taken by drilling into the center of the test piece, thus getting the core only. The right way would have been to drill from the skin to the center and use all the drillings for the analysis. By getting the core only, the possibly heavier percentages of combined carbon near the surface, when comparing high with low oxygen samples, with identical core results, would be lost. Hence I cannot agree to his conclusions, for the stronger metal shown as due to higher oxygen content might just as readily have come from other causes, such as lower silicon with higher combined carbon nearer the skin, etc.

I would take exception further to the statement that the cupola process is not an oxidizing process, or a meaning to that effect. In the cupola we oxidize almost as much silicon and manganese as in the air furnace or the open-hearth furnace. Ordinarily there is lost in the cupola 0.25 silicon, and in the other furnaces mentioned 0.30.

Then again, what about the deoxidizers, such as titanium, vanadium, aluminum? They strengthen the metal by removal of oxygen, or the very opposite of what is shown by some of the analyses given. Hence, as stated before, I think that the man who actually uses the pig iron by re-

melting it, while agreeing with Mr. Johnson that oxygen hardens the metal, will have to watch his work very carefully when using high-oxygen pig irons, to avoid other casting troubles which would make the work difficult to dispose of to a consumer.

HENRY D. HIBBARD, Plainfield, N. J.—Others besides foundrymen find much in this most interesting paper, but there are several points which may need further attention. The change in the method of occurrence of the graphite in the iron due to the oxide of iron contained seems to amply explain the increased strength obtained by the addition of the oxide.

Referring to p. 367, it would be well to know more about the iron of faultless analysis from which good steel cannot be made, as such iron would be a novelty to the writer and perhaps to some others.

Melting cast iron in a crucible may be done with a wide range in the final temperature, say from 1,250° to 1,550° C., and therefore one should know the temperature contemplated in the statement on p. 369 relative to remelting normal iron with the spongy No. 6 in order to better understand what happens. There is possibly a more or less clearly defined temperature below which carbon dissolved in iron will not reduce iron oxide, and if the temperature of the crucible melt is below that the result may be quite different from that when the temperature is above. This, if so, may explain the possible existence of carbon and iron oxide side by side in iron. What temperature does the author have in mind?

Regarding the question of the combination of sulphur and oxygen, temperature is also important. At steel-melting temperature, or say 1,550° C., the affinity of sulphur for iron is greater than for oxygen. At lower temperatures, such as those of the puddling furnace and refinery, sulphur is oxidized and eliminated to an important degree from the iron which is worked therein.

The reference on p. 382 to chilled cast-iron car wheels and conclusion No. 7 on p. 383 need some qualification. Manganese, in pig iron, which has been reduced in the blast funace with the iron, seems to increase the tendency of the carbon to remain in solution when the iron freezes, as the author states. But manganese added, after it has left the melting furnace, to molten iron intended for car wheels has the effect of decreasing the depth of chill by tending to throw the carbon out of solution as graphite. This has not so far been explained, I believe, but is true, and ferro-manganese, rather finely ground, is used regularly in car-wheel foundries to cut down the depths of chill of iron whose test shows too great chilling qualities.

HENRY M. HOWE, New York, N. Y.—I do not think we will keep to our belief that oxygen is *malum in se*. Mr. Johnson does not look at the oxygen as the cause of the matter—he looks at the oxygen as reducing

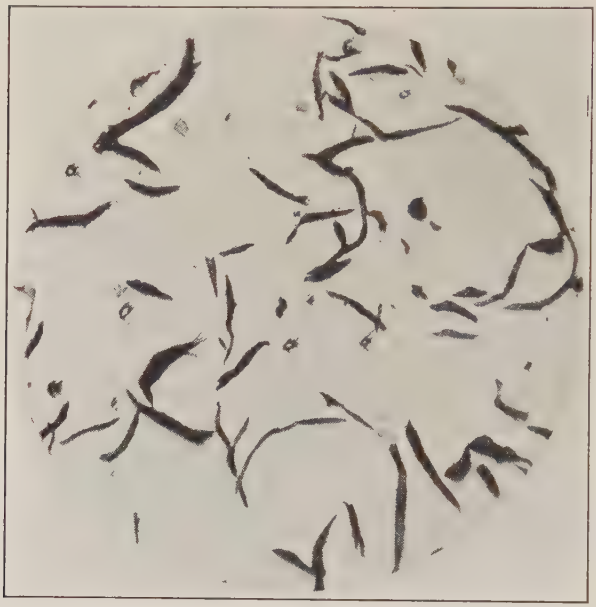


FIG. 30.—UNIFORM DISTRIBUTION OF GRAPHITE. × 100.

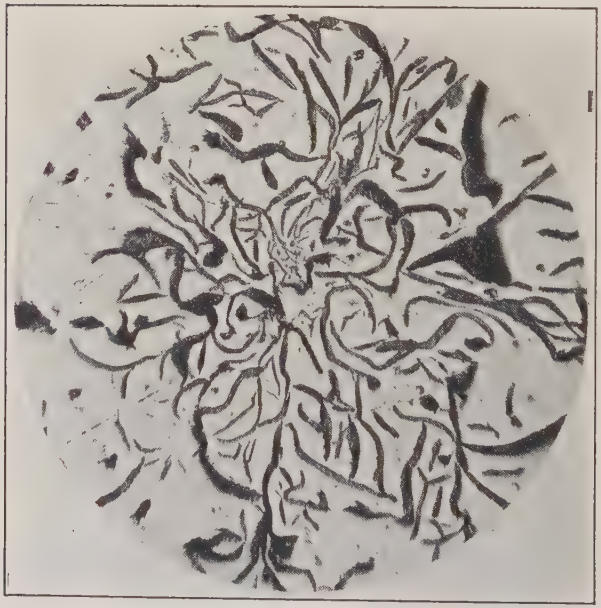


FIG. 31.—SHOWS NESTING OF GRAPHITE. × 100.

the size of the graphite flakes, so in spite of the harm the oxygen itself does, there is a net benefit in decreasing the size of the graphite flakes to a great extent.

C. H. Strand, Altoona, Pa.—I would like to ask Mr. Johnson what an uneven distribution of graphite, as shown in the accompanying photomicrographs, Figs. 30 and 31, indicates as to the quality of the cast iron. These two photomicrographs were taken on spots about $\frac{1}{8}$ in. apart, the nests of graphite occurring periodically throughout. Other cast irons show an absolutely uniform distribution of graphite. The opinion has been offered that this condition indicates a too low casting temperature. I would like to have Mr. Johnson's views on the matter, and whether he considers this condition to indicate a poor-quality iron.

William R. Webster, Philadelphia, Pa.—I will ask Mr. Johnson if he will make clear the difference between the so-called semi-steels produced from using slab in the pile of wrought iron or steel, and his steel. I believe that one is a low-carbon proposition and the other is a high-carbon proposition.

John H. Hall, New York, N. Y.—On reading Mr. Johnson's most interesting paper, it at first appears that the oxygen content of the iron is wholly or chiefly responsible for the differences in strength between the good and the bad irons of similar analyses. More careful consideration, however, leads one to modify that verdict very considerably.

For convenience, the more important tests given are summarized in the two tables herewith. From Table I we see that the differences in strength between the good and bad irons of the typical six given, are more or less in proportion to the differences in oxygen content. Table II, however, shows us that differences in oxygen content between original and deoxidized irons, much larger than the differences in all but one case in Table I, not only do not cause the reductions in strength that we should expect if oxygen alone causes the variations, but even result in considerable strengthening of the iron in many cases. Deoxidation with either aluminum or titanium appears to strengthen the No. 2 irons.

The only iron in Table II that shows reduction of strength after remelting with a deoxidizer, in enough cases to lead us to believe reduction of strength to be the rule, is the iron labeled "Medium 3." Why should this iron be so weakened by the elimination of oxygen, while the others are strengthened?

At first glance one is led to say it is because the amount of oxygen removed is the greatest in this iron. Yet comparing the decreases in strength resulting from deoxidizing this strong "Medium 3," with the differences between a weak and a strong "Medium 3" (Table I), we see that the reduction is insufficient to allow us to believe the oxygen is the only variable responsible for the weakness of the low-oxygen irons.

The weakening of this iron on remelting with deoxidizers, removing some 0.044 per cent. oxygen, is of the order of magnitude of the difference between the good and bad "Soft 1" (Table I), with an oxygen difference of but 0.018 per cent.

Thus the removal of oxygen by aluminum or titanium weakens the strong irons only in the case of the "Medium 3," and does not weaken this iron as much as we should expect, even allowing for any strengthening effect of titanium. We must look further for the complete explanation of the facts observed, and to my mind we find it in the total carbon content of the irons. All are near the probable location of the eutectic point, some 4.10 per cent. carbon according to Wittorff, and inspection of Table I shows that the greatest differences in strength correspond with the greatest variations in total carbon. This is especially the case with the "Medium 3" iron. The weak "Medium 3" contains 4.49 per cent. of carbon, and hence is well above the eutectic ratio.

Comparison of Weak and Strong Irons of Similar Analyses

Iron	O_2 Difference	Total C Difference	Excess Strength, Good over Bad, Pounds		
			Original	Remelted 1.25-in. Bars	2-in. Bars
Soft 1	0.018	0.06	475	525	2,500
High 2	0.018	0.11	800	850	2,050
Medium 3	0.056	0.39	1,350	1,400	8,150

Effect of Remelting Strong Irons with Deoxidizers

Remelt No.	Iron	Deoxidizer	O_2 Difference	Excess Strength, Pounds			
				Deoxidizing Remelt over Original		Deoxidizing Remelt over Plain Remelt	
				1.25-in. Bars	2-in. Bars	1.25-in. Bars	2-in. Bars
278 / 277	Low 2	Al	0.036	+ 50	−4,500	0	−3,700
279 / 277	Low 2	Ti	0.03	+500	.....	+450	+3,900
305 / 297	High 2	Al	0.02	+250	.....	− 50	+1,050
303 / 301	Medium 3	Al	?	−350	.....	−650	+1,600
306 / 301	Medium 3	Al	0.044	+150	.....	−150	−1,250
304 / 301	Medium 3	Ti	?	− 50	.....	−350	−3,000

Of the irons remelted with deoxidizers, "Medium 3" contains the most total carbon, 4.10 per cent., and is weakened the most by deoxidation. The irons strengthened by deoxidation contain carbon below the eutectic ratio. The conclusion is almost forced upon us that the effect of oxygen is marked only when the iron is near the eutectic ratio. A shifting of this ratio by increased oxygen content may throw a normally hyper-eutectic iron into the hypo-eutectic class. In this connection the fact that the weak irons of the typical six contain in every case more total carbon than the strong irons, is suggestive. Whether this shifting of the iron to the hypo-eutectic class is capable of producing the fine graphite formation shown, I am not prepared to state. I should suppose, however, that it would be.

The improvement in the strength of weak irons produced by remelting them with an admixture of "spongy No. 6" may also be explained in this manner. Mr. Johnson, judging by his remarks in paragraph 3, p. 357, seems to have seen that the influence of oxygen and that of high total carbon are jointly responsible for the facts he observed, but to have laid the heaviest emphasis on the influence of the oxygen.

It would be of great interest if Mr. Johnson would give us more data, as, for instance, the microstructures and more complete analyses of the irons remelted with deoxidizers, complete analyses, tests, and microstructures to show the effect of remelting "weak spotted irons" with "spongy No. 6," complete analyses, tests, and microstructures to show the status of the cold-blast charcoal irons referred to, analyses and tests of the electric-furnace irons, etc. The subject matter of the paper is so important and the opportunity for contributing greatly to existing knowledge so clear, that more ample data will inevitably be of the utmost value. As the paper now stands one is constantly headed off in following a line of reasoning by the dilemma which confronted Hosea Bigelow's campaign orator: "Tries pump again—no facs!"

Albert Sauveur, Cambridge, Mass.—Mr. Johnson has again placed those interested in cast iron and its properties under very great obligation by his well-planned and carefully conducted investigation of the puzzling fact that some irons—the good ones, as he terms them—always give strong castings, even after repeated remelting, while others, of *apparently* the same chemical composition—the poor irons—always yield weak castings. Mr. Johnson claims that the presence of a larger proportion of oxygen in the good irons is responsible for this great strength and he contends that his experiments prove his claims beyond any possible doubt. While at first sight it may appear that he has demonstrated his contention, upon closer scrutiny and thoughtful reflection doubt arises as to his having discovered the real cause of the phenomenon under investigation. The author shows a difference of a few hundredths of 1

per cent. of oxygen between the weakest and the strongest irons, otherwise of similar composition, so far as analysis of cast iron generally goes, and we are asked to believe that so very slight a difference in oxygen content accounts satisfactorily for the much greater strength of the good irons. He contends, for instance, that increasing the percentage of oxygen from some 0.02 to 0.1 per cent. is responsible for an increase in transverse strength from 3,000 to 5,000 lb. Turning to the photomicrographs illustrating the structure of the various irons, we see clearly why the good irons are stronger; it is evidently because their graphite occurs in small, and often rounded, particles, instead of occurring, as it does in the bad irons, in the form of long, curved and sharp plates or sheets. It is for a similar reason that malleable cast iron is always stronger than gray iron of like chemical composition. We are naturally reluctant to believe, however, that so marked a difference in the mechanical behavior of graphite is caused by a difference of a few hundredths of 1 per cent. of oxygen between the two classes of cast iron. To accept this explanation, we certainly must have more conclusive evidences than those offered by the author.

Granting that Mr. Johnson has satisfactorily shown that good irons contain more oxygen than poor irons, and that the furnace conditions necessary to produce strong irons are generally, also, those which lead to less perfect deoxidation, the inference that the greater strength of the good irons is actually due to the presence of a slightly greater percentage of oxygen is very far from conclusively demonstrated. Were it so, it should be a comparatively easy matter to increase the strength of poor irons by oxidizing them, and to decrease the strength of the good irons by deoxidizing them, and it is a significant fact that the author's attempts in both of these directions have failed.

The results published in Tables I and II are not convincing, or even always consistent. The average strength of the C High 2 iron, for instance, containing 0.038 per cent. oxygen, is 3,800 lb., whereas the average strength of the E Med. 3 iron, with 0.065 per cent. oxygen, is but slightly greater, namely, 3,950 lb. In Table II, the irons low in oxygen appear to be as strong as, if not stronger than, those containing more oxygen.

Summing up, the utmost that may be granted is that the author has shown satisfactorily that good irons generally contain more oxygen than poor irons. He has not demonstrated that poor irons can be made stronger through the introduction of oxygen, or that good irons are made weaker through the removal of some of their oxygen. The fact that normal iron (containing little oxygen) is made stronger by melting it with "spongy No. 6" (containing more oxygen) does not prove that oxygen is actually the cause of the increased strength. It merely points to some

qualities, inherent to the spongy No. 6 and making for strength, being imparted to the resulting mixture.

Bradley Stoughton, New York, N. Y.—I really think that Mr. Johnson's analyses show definitely more oxygen in some cases than in others, and this offers much more evidence than the absence or deficiency of oxygen by inference. Of course, I do not know but that Dr. Moldenke has made these analyses, too, but I was simply taking what he said to be limited by as much as he said.

In regard to Mr. Hibbard's remark about faultless iron, as far as the analysis is concerned, not making good steel, I have had that experience a number of times. There is a sort of iron which comes from the blast furnace out of which you cannot make good steel. It has been my misfortune to note that many times. We have never been able to find out, in the analyses, anything that is wrong with that steel, but I have seen the percentage of second-class rails go up from 3 to 22, and it was an unhappy time for me, because I was supposed to be responsible in any event.

It finally got so bad in a large works with which I was connected that a practical man there could tell this iron as it ran from the furnace—it had a peculiar appearance—and in order to prove that it was the iron which was not satisfactory he would take a ladle of iron every once in a while and send it directly to the converter to convert it into steel, without its passing through the mixer, the analysis, of course, being correct, so that we were able to do that. In every case we had very bad results with the steel. Furthermore, I had to convince the men having charge of this steel that if a certain number of ladles of this kind of iron went into the mixture we would have unsatisfactory results, and in each case we would find the percentage of second-quality rails went up.

The analysis of that iron was all right, but it had a peculiar appearance to it, which none of us could tell except the one man to whom I have referred, and finally we used to keep him at the blast furnace a greater part of his time trying to spot that iron and prevent it from going into the mixer and being made into steel, much to the disgust of the blast-furnace man, who was positive, of course, that it was the fault of the steel mill.

It seems to be admitted by all of us, from what I have heard said, and it seems very evident to me, that these irons high in oxygen, but otherwise practically the same, will have a better structure than the irons which do not have oxygen in them. It seems to me, even if the oxygen in itself is harmful, provided the net result is good, there should be no objection to the oxygen. Certainly we agree that the iron with graphite in large flakes is likely to be, as Mr. Johnson certainly indicates, very strong, and is stronger than the iron in which the graphite flakes break up easily.

The fact that the amount of oxygen is very small does not seem to

me to be material at all. We do not know what form the oxygen is in. One-thousandth of 1 per cent. of oxygen may mean quite a good deal of iron oxides or some other oxides, and we have always had that augmented with a small amount of phosphorus. If you remember, the small amount of moisture in the blast furnace was judged to be ridiculously small to produce any beneficial effect on the work of the furnace. Consequently our German friend patented the idea and decided that the Americans did not know what they were talking about in blast-furnace practice until they were obliged to admit that their theory was defective in some way.

It has occurred to me, and I find in conversation with others it has occurred to several persons, that possibly the oxygen may have an influence in throwing the eutectic point to the right in the diagram, and several have mentioned that particular thing to me. I do not see any evidence to confirm that view, of course, except by inference, and if this occurred it would have an effect, perhaps, on the oxygen, and might have a marked effect on the quality of the iron.

William Campbell, New York, N.Y.—From looking over Mr. Johnson's figures and photographs we must be led to the conclusion that the difference in strength depends mainly on the forms of graphite. In examining quite a number of samples of cast iron, we find that there is certainly a difference in the way in which the graphite comes out. Some of it comes out directly from the liquid, whereas other forms of it appear after the metal is solidified. Looking over the different iron-carbon diagrams, of which there are many, we can get some evidence of this; if the later determinations of Wittorff and similar workers are right we have the formation of cementite just above the freezing point, which afterward breaks down into the graphite. The natural conclusion is that the graphite which forms from decomposition of cementite will be finer than that which comes out of the melt. It may be that the small amount of oxygen present in the iron may be the factor which swings over from the stable diagram on the one hand to the metastable on the other, and in one case we get mainly the primary graphite and in the other case we get the secondary graphite.

Ralph H. Sweetser, Easton, Pa.—Possibly some experience I had in Canada may throw some light on this "good iron that makes bad steel." I happened to be at the blast-furnace end, and my position was not any more cheerful than that of Mr. Stoughton. At the Lake Superior Corporation we made iron for the Bessemer department, and it was made wholly with Pocahontas coke. There were times when we had the very best of iron, it would be what we would call in the foundry No. 2, or 2X iron, but the graphite was very high, and the kish that would come off from the iron would collect everywhere, yet they claimed at the steel plant that the iron did not make good steel. The analysis showed just

exactly what they asked for in silicon, sulphur, and manganese, but they had trouble in the mill, and they blamed it, of course, on the blast furnace. We found large flakes of this kish, that were almost transparent, and when they were analyzed we found that they were a combination of silicon and carbon. I think there was just exactly one atom of silicon and one atom of carbon in some of the flakes we separated.

Possibly the making of iron with a very soft coke makes what we call a "rich iron."—the steel men said "Your iron is too rich." The air would be full of kish, and when that iron was cooled down in the ladles the top of the ladle would be covered with a thick layer of kish.

Henry D. Hibbard.—I am not satisfied to let the matter rest so. Mr. Stoughton's experience with those irons was when he was much younger than he is now. It is noticeable that his trouble with the apparently good iron of which good steel could not be made was in connection with the Bessemer process, which of course does not have as wide a scope in the materials used nor admit of variations in anything like the degree that the open-hearth process does.

Another thing is that in plants where only one product, like rail steel, is made they do not have much variety of methods or experience with other steels. Then when the regular practice with the regular stock does not yield the required result they are more or less at sea and may blame the stock when some change in manipulation might have made all right. Therefore, while I do not consider that Mr. Stoughton's experience proves the author's statement, I think it would be well if he would give us some analyses of those irons and say what he was making of them.

F. L. Grammer, Leesburg, Va.—I think Mr. Johnson has correctly attributed to temperature the position of chief factor in determining whether the hearth has the maximum reducing tendency and I also am inclined to believe that the chief source of oxygen is to be found in the descending ore. Nevertheless, there are other influences than temperature and there are other possible sources of oxygen than the ore. Perhaps the following may be of interest, even though the writer believes the first two influences are paramount.

Under Carbon Absorption, p. 372, I would like to add two more known facts:

3. The larger the amount of surface offered by a unit weight of carbon the shorter the time required for its solution in a given weight of molten iron. This is a relation governing all solution if interstices are not too minute for liquid to enter.

4. Solution as it approaches saturation proceeds more slowly.

While in discussion of oxygen contents we are not so much concerned with the first two postulates of Mr. Johnson as with the last two, these

added factors if considered will be seen to be the product of conditions assisting deoxidation.

Let us assume arbitrarily for conciseness that the metal as it drops into the bath of iron through the slag contains 2 per cent. carbon in solution and that it issues from the furnace with 4 per cent. carbon content. The 2 per cent. increase would be obtained by the bath dissolving carbon from incandescent fuel which penetrates below the floating molten slag into the molten iron. At casting times the solid fuel goes to the hearth bottom. That it does this after the first flush of cinder is unusual.

We are told that recently some of the large steel companies are giving their blast-furnaces crushed coke, rather than as it formerly came from beehive ovens. The motive probably is better distribution and regular working. Such coke, offering more surface per unit of weight, should produce a metal with lower oxygen content, other conditions being constant, than one with larger lumps. The intermixture in the bath is more complete.

The Scandinavian mystic and metallurgist, Swedenborg, I am told, first announced that spheres of equi-diameter tetrahedrally piled exhibited a ratio of voids of 29 per cent. and aggregates 71 per cent. So the amount of space in the hearth occupied by the fuel (if of equal spherical size) is practically independent of the size of the lumps. The amount of surface offered for attack by the molten material (occupying the voids) increases as the diameter decreases.

The incandescent fuel may not penetrate the slag and iron bath because of the pressure of the superincumbent stack contents; usually we believe this pressure on both is due chiefly to the weight of a dome-capped cylinder of incandescent fuel extending from a hemisphere, at the flare of the bosh near the tuyères, downward. The weight of material in the stack is borne by the bosh (on which stock momentarily arches) and by friction and ascending gases.

This dome-capped cylinder of fuel is taller after casting than before casting.

Taking the bath of iron and slag at cinder-notch level on the first flush, it might be fair as furnaces are built to say that the maximum fuel heights of dome-cylinder would be in coke furnaces using Lake ores about 9 to 13 ft. and in charcoal furnaces about 6 to 10 ft. Disregarding penetration below the molten surface, we may say for anthracite, 10 ft.

We will call the specific gravity of anthracite 1.6; coke, 1.0; and charcoal, 0.4.

Then this dome-cylinder will give a fuel pressure on top of the molten surface as follows:

Anthracite	$10 \times 1.6 = 16$
Coke	$(9 \text{ to } 13) \times 1.0 = 9 \text{ to } 13$
Charcoal	$6 \text{ to } 10 \times 0.4 = 2.4 \text{ to } 4$

If this is the pressure which controls penetration into metal, it will be seen that charcoal has nothing like so long a contact nor so deep a penetration into the metal bath as coke has, not to mention anthracite. Charcoal just before casting or before the first flush probably does not pierce the slag. Oxygen in the bath thus is less likely to be removed.

One may protest that we are not dealing with spheres nor even with sized particles, and that our 2 per cent. carbon is arbitrary. Many will question this method of stock settling. They will say it is due to the underpinning by solution and combustion of a fuel column resting on the hearth bottom and not by constant renewal of a fuel arch springing from the bosh batter commencing above the tuyères. A study of stock-line subsidence at casting time will prove that fuel penetrated the molten mass but slightly.

Our picture is probably more closely followed by narrow crucibles using charcoal than by coke crucibles of greater width and weaker dome. The truth is somewhere between. We have all seen coke at the bottom iron notch: it should always be in front of the tuyères. But we have heard of metal being partly Bessemerized in the blast-furnace crucible owing to fuel descent being delayed. It is certain that a very large percentage of the weight of stock is borne by the bosh and the stack and the wind.

We think most would concede that charcoal would dissolve in iron more rapidly than coke or anthracite, not to mention graphite. It is true that in recarburizing steel the solution of added coke is very rapid, but at blast-furnace temperatures and saturation of metal the surface of fuel exposed and the time of contact rather than the nature of fuel govern carbon contents. The final carbon increments are slow of growth —certainly if these oxide products are with such difficulty removed; time and full surface exposure where greatest in crucible should be associated with lowest oxygen.

This picture is timidly sketched in the hope that some may see additional possible explanations besides these offered by Mr. Johnson.

CO_2 reacts on C at certain temperatures depending on preponderance of fuel and its nature and velocity of passage of gases. We would expect this temperature to be lower with a soft, cellular fuel like charcoal, which occludes, than with a dense, hard fuel (resembling incombustible graphite) like anthracite. The zones of combustion per unit of blast will be larger in the case of hard and dense fuels than with porous fuels. This seems to be an additional reason other than slag composition why coke and anthracite furnaces need high temperatures of blast compared to charcoal practice.

Magnetites enter the hottest zone with more oxygen than other iron ores save a few specular ores and so it seems reasonable to expect more oxygen in metal bath from such mixtures.

Owing to low ash in charcoal, in such a furnace the slag is practically

all complete as it drips into the crucible. In a coke furnace, a large percentage of the slag comes from ash which is released as combustion in the hearth proceeds. It unites with the excessively basic product descending from the bosh.

A coke furnace requires reserve heat to fuse a belated slag completion, and to take care of a slag whose volume is excessive, or basicity and temperature are high, in order to prevent sulphur entering the metal.

A unit weight of oxygen in entering a hot-blast furnace occupies thrice the volume of a similar weight furnished the charcoal furnace. Besides the increased *velocity* of gases incident to excess heat in coke furnaces, the velocity is also larger because of the larger weights of air (rapid driving) employed.

The cold-blast charcoal furnaces offer converse conditions to those cited as obtaining in coke practice. Whether the oxygen in metal has its origin in descending charge or entering blast, the two furnace operations are so diverse as to lead one to expect differing products.

Thinking of occlusion, one would have expected the higher-pressure coke furnaces to show higher gas contents in metal.

Throttling the downtake increases the gas tension in the furnace very markedly, preserving at the same time quantities and velocities of blast. It might be interesting to note whether metal made under such conditions exhibits change as regards gaseous contents.

Antimony, unlike phosphorus, with which it is often grouped chemically, seems to increase total carbon contents in pig.

I analyzed some kish scooped up from the ground in 1894 at the Edgar Thomson Furnaces and found it contained a large per cent. of silicon, as Mr. Sweetser cites.

We had a case of "dead iron" at Duquesne in 1897 in which a lowering of the temperature of the blast accompanied by the reduction of the burden was followed several hours later by a reduction in the per cent. of seconds of the rails.

There is quite a widespread belief that iron made from magnetites is better than that made from hematites, and we rather expect those who have had the misfortune to use these irreducible ores to investigate the possibilities of the superior value of their product as compared with hematite metal.

In addition to investigations pursued to ascertain the influence on oxygen content of gas throttling, magnetites, crushed coke, and moisture elimination, we also think it would be of interest to ascertain whether metal made from furnaces with 20 tuyères as compared to furnaces with 10 tuyères has a different oxygen content.

J. E. Johnson, Jr.—Replying in part to Dr. Moldenke's criticism, I would call particular attention to an analysis on p. 346 of my paper in which it is shown that the variation of combined and graphitic carbon in these

irons is exactly the opposite of what it should be if the accepted theory were correct; that the strength increases primarily with the quantity of combined carbon and decreases with the quantity of graphite. This latter is a perfectly logical and obvious conclusion if we disregard variations in the size and shape of the graphite particles. I have therefore taken some pains to analyze these results so as to show that the strength varied in these six irons in the opposite direction from what these considerations would indicate, and this seems to me to disprove absolutely any contention based on the idea that the variations in the combined carbon are responsible for these results. This absolute contradiction of this obvious and natural conclusion indicates that there must be some cause for the results totally independent of this consideration. This cause, I believe, will be found by any one who examines the photomicrographs critically, in the size and shape of the graphite particles, for there is no important cohesion between graphite and iron, and each flake of graphite is therefore neither more nor less than a cut in the iron of area equal to that of the graphite plate. To carry this idea to its ultimate conclusion we need only to turn to the salamanders which all furnacemen have taken from the hearths of their furnaces and which contain flakes of graphite a square inch or more in area and can be broken apart along these flakes with the fingers. What a flake of this size would do to a $1\frac{1}{4}$-in. test bar is obvious and is an index of the importance of the size of the graphite. I have tried to make it clear in the paper that it was through this control over the size of the graphite particles that oxygen exerted its principal influence.

In regard to Dr. Moldenke's statement that he had made many hundreds of oxygen determinations, I understand that these were made by a wet or acid method, and I do not consider these methods reliable for the determination of oxygen, since almost every action of solution is an oxidizing one in one form or another, and for that reason oxygen may be introduced into the very substance whose oxygen content is sought to be determined.

The determinations on which my paper is based were made by the Ledebur hydrogen method as modified by Cushman and by ourselves to suit the conditions arising from the presence of so much more carbon than is present in steel. So great an amount of care and time was put upon these determinations, such satisfactory blanks could be run with them and results could be so carefully checked, that it is impossible to escape the conclusion that the oxygen which we found in the form of water came from the iron and from the iron alone.

I would like to call attention here to the fact that much of what Dr. Moldenke himself has written on this subject can be as well explained on my hypothesis as on his, for he has repeatedly spoken of getting the bath too hot and burning it. What really happens, in my judgment, is that when the bath becomes too hot the oxygen reacts with the carbon and

silicon present, so passing out of the metal and lowering its quality in the way which Dr. Moldenke attributes to "burning."

In regard to the allusion in the paper to the reducing tendency of the cupola as compared with the oxidizing tendency of the air furnace, these terms are relative and were used by me solely in a relative sense, and I think it will be admitted by Dr. Moldenke that this is justifiable, because the oxidizing influence of the blast in the cupola is partly offset by the fuel bed below, whereas in the air furnace the oxidizing tendency of the flame is much greater and is not offset by any circumstances whatever. Therefore, I think it will be admitted that the likelihood of the introduction of oxygen in the air furnace is much greater than it is in the cupola—the idea which I intended to convey.

In reply to Mr. Hibbard, I can only say that if he will take a trip around among the steel works and obtain the confidence of the metallurgist or the steel mill superintendent, either of them will not only tell him that the quality of the steel is seriously affected by an unknown quality of the iron from which it is made (not shown by the analysis of the latter), but that the situation so produced is a serious one against which they have constantly to be on their guard. This, as far as I can find out, has passed from the realm of dispute into the realm of accepted facts among those who know.

In answer to W. R. Webster, I would say briefly that the difference between high-oxygen cast iron and semi-steel is briefly that the high oxygen content reduces the graphite particles to a small fraction of their area in ordinary cast iron, while semi-steel lowers the graphite present as a whole by diluting the relatively high carbon cast iron with relatively low carbon steel. This makes a less quantity of graphite to exert an injurious influence and there may be a certain oxidation of the steel scrap, and consequent introduction of oxygen into the bath, with beneficial result, but this does not compare with the results to be obtained in an iron with normal or slightly subnormal carbon which contains a sufficient quantity of oxygen to suppress the formation of flat flakes of graphite. The iron produced by the latter method is stronger and capable of more satisfactory handling both in the foundry and in the machine shop.

In reply to C. H. Strand, I will freely confess that I cannot answer his question positively. In a general way I should be inclined to connect the formation of graphite nests, such as shown by his photomicrographs, with the formation of windrows of graphite, the formation and cause of which were pointed out in my paper, The Effect of High Carbon on the Quality of Charcoal-Iron.[3] I should expect that these graphite nests indicated an iron rather high in carbon and very low in oxygen, consequently, a weak iron, but not being familiar with the phenomenon he points out in the shape in which he shows it, I cannot answer positively.

[3] *Trans.*, xliv, 314 (1912).

In answer to John Howe Hall, I admit freely that the data submitted in the paper are insufficient to enable us to attribute a given increase in strength to a given increase in oxygen, and it is true that some of the results on deoxidation were contradictory. At the same time the impression formed as a consequence of the more careful and more reliable of these results, that is, the ones later made, was that they did result in a considerable weakening of the iron. This is particularly true with aluminum, whose effect we understand much better than we do that of titanium.

In regard to Mr. Hall's attempt to connect up the quantity of carbon with the strength of the iron, I tried to make it clear in my paper that the conditions which led to high carbonization were largely similar to those which led to thorough deoxidation, and referring back to my previous paper I intended to make clear my opinion that both of these conditions contributed to weaken some irons. At the same time, a careful analysis of the carbons and oxygens shown in the original paper, and also in the additional data put on the blackboard at the meeting and given below, will show that there is no unvarying connection between total carbon and strength, but that almost as much evidence can be obtained for one side as for the other. This, however, is very far from being the case with the oxygen content, since this in every case is high with the strong iron and low with the weak iron. It is very doubtful if, even with the additional data here presented, any quantitative relation between oxygen content and strength, in the presence of so many other variables, can be established, but the qualitative effect is so clearly shown as to be obvious to all save those who do not desire to see.

			Mn	P	S	C	Si	O	Breaking Strength Pounds
A	4/20/13	10 p. m....	0.76	0.134	0.016	4.06	2.56	0.050	3,450
	11/1/13	10 a. m....	0.70	0.160	0.016	3.80	2.58	0.009	2,400
B	10/20/12	4 p. m....	0.30	0.150	0.020	3.75	1.17	0.068	3,700
	2/10/13	10 p. m....	0.46	0.119	0.015	3.90	1.00	0.065	3,400
									3,500
	10/14/13	4 a. m....	0.26	0.156	0.022	3.82	1.10	0.012	2,350
C	2/10/13	10 p. m....	0.40	0.119	0.015	3.90	0.70	0.055	3,400
									3,500
	3/11/13	4 a. m....	0.70	0.112	0.025	3.98	0.61	0.064	3,700
									3,825
	3/12/13	10 a. m....	0.72	0.106	0.020	3.76	0.60	0.061	3,950
									3,200
	8/10/13	4 a. m....	0.37	0.134	0.021	4.01	0.65	0.010	2,500
	11/9/13	4 a. m....	0.38	0.160	0.019	4.14	0.60	0.009	2,500

In reply to the remarks of Mr. Stoughton, I have only to express my thanks for the confirmation of my statements and views which he so clearly and comprehensively furnished.

In regard to Professor Campbell's remarks, it is most unfortunate that lack of time prevented his explaining in detail certain possibilities based on the latest developments of the iron-carbon diagram which he was good enough to explain to me privately. Through his kindness I am able to present here three possibilities based on the work of Guertler, who founds his conclusions upon the experiments of Wittorff and Ruff, and one based on Upton's iron-carbon diagram. According to Guertler we have:

I. In the stable condition above 1,360° C. we can have liquid + graphite.

At 1,360° C. the reaction occurs,

$$\text{Liquid} + \text{graphite} = Fe_4C$$

and in alloys with 4.4 per cent. carbon down to 3.8 per cent. we find Fe_4C crystallizing out from the melt.

At 1,180° C. we have the eutectic of austenite + Fe_4C.

At 1,110° C. Fe_4C breaks up into austenite + graphite.

At 750° C. the eutectoid of ferrite and graphite forms.

These are equilibrium conditions which are reached only with extremely slow cooling.

II. With more rapid cooling the reaction at 1,360° does not occur. Graphite separates from the melt and at 1,155° we get the eutectic of austenite + graphite.

III. With more rapid cooling Fe_3C separates out from the melt and at 1,135° C. and 4.3 per cent. carbon forms a eutectic with austenite; then at 710° C. we have the eutectoid Fe_3C + ferrite, commonly known as pearlite.

According to Upton's diagram white irons are supersaturated solid solutions which are breaking down; and the stable constituents are austenite + graphite above 1,095° C.; austenite + Fe_6C or Fe_6C + graphite below 1,095°; and at 800° Fe_3C appears, while Fe_6C disappears. Fe_6C and Fe_4C have yet to be distinguished under the microscope.

From the point of view of practical value two possibilities according to these diagrams stand forth prominently:

1. Graphite can separate directly from the melt.

This is contrary to the claims of Professor Howe and Professor Sauveur, but is abundantly supported by the facts, while their contention that this action can only take place through the prior formation of cementite is completely negatived by facts of practice which are matters of common knowledge to furnacemen.

2. After solidification various possibilities are presented according to both of these investigators. These possibilities indicate that in cer-

tain areas of the diagram a very slight change in conditions may cause a complete change in the resulting product. Professor Campbell's valuable suggestion is, as I understand it, that a small quantity of oxygen may be the determining factor in controlling which one of these conditions arises, and as a matter of great practical and industrial importance I know it to be true as the result of abundant investigation that not only the shape of the graphite is greatly influenced by a change in the oxygen content of the iron, but that the structure of the iron itself appears to be fundamentally altered by the presence or absence of this element. When it is absent certain irons tend to form the iron-carbon eutectic, whose effect is absolutely detrimental from every point of view, while the presence of oxygen appears to suppress this structure altogether and forms instead a network structure concerning which the microscope furnishes the presumption that it should possess superior strength, a presumption abundantly borne out by physical tests. It would seem that scientific metallurgists interested in this subject may find interesting work for years to come along this line.

Turning now to the practical point of view and in particular answering the criticism of Professor Sauveur that our tests do not carry conviction that the oxygen is the cause of the change because, he says, remelts with spongy No. 6 do not prove our case, I will furnish additional evidence developed since this paper was written.

A short time ago I took a heat of coke iron containing about 1 per cent. silicon, from which test bars were made as it came from the cupola. This heat was put into a Stoughton side-blow converter and had the silicon blown out of it at a temperature as low as we could safely hold it. The blow was stopped before much of the carbon had gone, and the resulting metal, which was a wild white iron full of blow-holes, utterly rotten and worthless, as shown by samples which we poured, was mixed with an equal weight of 2 per cent. silicon iron from the cupola, from which test bars were made previous to mixing. From the mixture additional test bars were made. These included $1\frac{1}{4}$-in. rounds, as with the two kinds of iron from the cupola, and also 2-in. square bars. These were broken by Tinius Olsen & Co. in Philadelphia. The results on the cupola irons of both heats ran from 2,000 to 2,500 lb., while the bars made from the mixture of converter metal and cupola metal broke at from 3,900 to 4,200 lb. The 2-in. square bars of the latter mixture broke at from 19,000 to 24,000 lb., the latter figure considerably in excess of the most rigid requirements of the purchasers of Southern and Eastern charcoal irons. It will be seen that by mixing the siliconless converter metal with 2 per cent. silicon metal we restored the iron practically to the analysis of the original melt in everything save oxygen, and this seems to me to furnish as conclusive a proof as the most exacting can require that it is oxygen and oxygen alone which produces these results. This iron not only had the strength, but also had the close grain, the high density, and

the great chilling power of the best grades of charcoal iron, from which it could not be distinguished by the most careful examination. Photomicrographs of this iron, etched and unetched, Figs. 32 and 33, are presented herewith on the same scale as those in the original paper. The similarity of this iron to those charcoal irons will not, I believe, be denied by any one, and I believe those familiar with the microstructure of malleable castings will be struck with the similarity of the unetched specimen to the structure of malleable iron.

It is suggested in the paper that the explanation for the action of

FIG. 32.—COKE IRON + OXYGEN. UNETCHED. × 100.

oxygen is that it reverses the points of graphite formation and solidification, which ordinarily occur in the order stated, but which in these irons containing oxygen, in my opinion, occur in the opposite order, as shown by the suppressed condition of the graphite in the photomicrographs. This opinion receives confirmation from a direction which did not occur to me at the time the paper was written. It is a matter of common knowledge among users of high-grade irons that strong irons of the kind they desire have low shrinkage, while poor irons have high

shrinkage, and the shrinkage test is actually used as one of the determining factors in the purchase of iron.

Now, if the foregoing explanation be correct it explains this fact also, because if graphite formation be delayed until after the partial solidification of the iron matrix, then obviously when the graphite does form the increase in volume which its formation involves will expand the partly solidified iron, and this expansion will partly offset the subsequent contraction due to cooling, so that the net shrinkage of the iron will be small. If, on the other hand, the graphite separates out while the iron is still

FIG. 33.—COKE IRON + OXYGEN. ETCHED HNO_3. × 100.

perfectly liquid, this action obviously does not occur. The contraction due to cooling is not offset by this expansion due to the formation of graphite, and the resulting shrinkage is greater than in the other iron.

Thus, this explanation agrees with another important factor, well established by practice, which seems to me to increase the probability of the correctness of the explanation very materially.

In the commercial aspect: While producers of coke iron ignore the fact, the manufacturers of high-grade castings pay premiums of from

$10 to $25 per ton for irons having these qualities. They do this not through inertia, not through ignorance, but only after thorough and convincing trials of coke iron as a substitute have demonstrated beyond doubt that they cannot obtain the results that can be obtained with these other irons. There is not only one such foundryman in the country, there are scores.

These expensive irons do not differ chemically in any way whatever from coke irons of similar analysis except in regard to oxygen.

Admitting that we have all, myself most strenuously with the others, until recently, believed that oxygen was a serious detriment to the quality of iron, have we any objective evidence whatever to support this view? Admitting the old idea that a difference in oxygen is accountable for these differences in quality, does not the view that its presence is responsible for the improvement agree far better with the actual facts than the view that its absence causes these differences? Are we not, therefore, entitled to lay aside our preconceived ideas, our prejudices, our beliefs and our theories, in order to accept the concordant results of a vast mass of uncontroverted and, I believe, uncontrovertible evidence of which this paper furnishes only a small part, and against which the only objection urged is that it is new?

The Surface Decarbonization of Tool Steel

BY J. V. EMMONS, CLEVELAND, OHIO

(Pittsburgh Meeting, October, 1914)

It has long been known that the outside skin of tool steel frequently exhibits properties widely different from the interior of the mass. Within the last few years it has been generally recognized that many of these properties were due to surface decarbonization. The origin and effects of this altering of the composition have been but little studied.

C. M. Johnson[1] arrives at the conclusion that, in annealing, "the most active agent in surface decarbonization is the rust or scale (present on the surface), which is actually reduced to metal at the expense of the carbon in the steel to which it adheres." W. H. Hatfield in his admirable paper, The Chemical Physics Involved in the Decarburization of Iron-Carbon Alloys,[2] has given us the most tenable theory of the removal of carbon from iron.

While Mr. Hatfield's work has been done principally in connection with the annealing of white cast iron to produce malleable castings, it is frequently applicable to the decarbonization of tool steel. The following equations which he proposed have been fully borne out by the author's experiments:

$$\left.\begin{aligned} 2Fe_3C + 2CO_2 &= 3Fe_2 + 4CO \\ 2CO + O_2 &= 2CO_2 \\ 2CO_2 + 2Fe_3C &= 3Fe_2 + 4CO \end{aligned}\right\} \text{Continuous}$$

It is the purpose of the author to show, in a quantitative manner, the effect of various treatments in eliminating the carbon in tool steel. Some of the phenomena of the decarbonized surface will also be described, as well as the practical problems they present to the manufacturer and user of tool-steel products.

A $\frac{1}{2}$-in. round bar of crucible steel having the following composition was selected: C, 1.15 per cent.; Mn, 0.31; P, 0.022; Si, 0.16; and S, 0.018 per cent. A microscopic examination having revealed a slight decarbonization already present, the bar was turned down to $\frac{3}{8}$-in. diameter to remove all traces of decarbonization. The bar was in

[1] *Journal of Industrial and Engineering Chemistry*, vol. i, No. 7, p. 459 (July, 1909).

[2] *Journal of the Iron and Steel Institute*, vol. lxxix, p. 242 (No. I, 1909).

the annealed condition, the structure consisting of finely divided lamellar pearlite, with many small grains of globular cementite. The bar was cut into test pieces of about 2 in. in length. The heating operations were all conducted in an electric resistance furnace of the tube type. The temperatures were regulated by means of a Le Chatelier thermo-couple.

Johnson having stated that he found a slight decarbonization upon annealing tool steel in hydrogen, this was first investigated: A test piece was placed in the furnace cold and gradually raised to a temperature of 1,800° F. in a current of carefully purified and dried hydrogen. After holding at this temperature for 3 hr. the current was shut off and the

Fig. 1.—100 Diameters.

piece was allowed to cool slowly in the furnace. On removal from the furnace, the test piece had a dull-gray crystalline appearance. Microscopic examination revealed no sign of decarbonization. A second test piece was placed in the furnace in the same manner as before, held at 2,130° F. for 6 hr., and then allowed to cool slowly in the furnace. This piece exhibited the gray crystalline surface in a still more marked degree. A microscopic examination revealed a coarse network of cementite extending to the extreme edge of the cross-section, giving conclusive evidence that no decarbonization had taken place. (See Fig. 1.) This proves that hydrogen is not a factor in the elimination of carbon from steel.

The part taken in the decarbonization of cast iron by iron oxide has been investigated by Wüst.[3] His conclusions are that ferric oxide, at the higher temperatures, gives off a part of its oxygen, which combines with the carbon in the iron to form CO. The CO still further reduces the ferric oxide to ferrous oxide, being itself oxidized to CO_2. The CO_2 again reacts with the carbon in the iron according to the equation $CO_2 + C = 2CO$, the process being continuous.

There is no doubt that these reactions are very closely duplicated in the annealing of tool steel. The bars are always covered with more or less hammer scale and rust, and are packed in a closed pipe or box. The principal difference is that, as there is no free carbon normally present in tool steel, the elimination of the carbon is probably accomplished according to the equation:

$$CO_2 + Fe_3C = 3Fe + 2CO.$$

In order to confirm these reactions, and also to determine the rate at which they proceed at various temperatures, the following experiment was carried out:

Pieces from the bar above described were annealed in a strong current of pure, dry CO_2, at various temperatures. The furnace was brought to the required heat, the pieces placed in it, held at the heat for 3 hr., and allowed to cool slowly in the furnace. Microscopic examination was then made of the cross-section, and the depth of decarbonization measured by means of a micrometer eye piece. For greater ease and accuracy, the decarbonization was measured only from the surface to the point of supersaturation, or to the point where free cementite first made its appearance. This point is much more easily determined than the point where the interior is first affected by the elimination of carbon. Each figure of the depth of decarbonization given below is the average of four measurements made at different points of the cross-section.

Piece No.	Temperature, ° F.	Depth of Decarbonization, Inches
1	1,350	0.000
2	1,400	0.007
3	1,500	0.009
4	1,600	0.018
5	1,700	0.025
6	1,800	0.034
7	1,900	0.053
8	2,000	0.072

[3] *Metallurgie*, vol. v, No. 1, p. 11 (Jan. 8, 1908); *Stahl und Eisen*, vol. xxiii, 2, No. 20, p. 1136 (Oct. 15, 1903).

A photomicrograph of the decarbonization of piece No. 3 is shown in Fig. 2.

A micrograph of piece No. 6 is shown in Fig. 3. The structure of piece No. 8, showing the outer portion of the decarbonization only, is illustrated in Fig. 4.

These results indicate that CO_2 gives up to the carbon very readily one of its atoms of oxygen. When the results are plotted in the form of a curve, with the depth of decarbonization as abscissæ and the tempera-

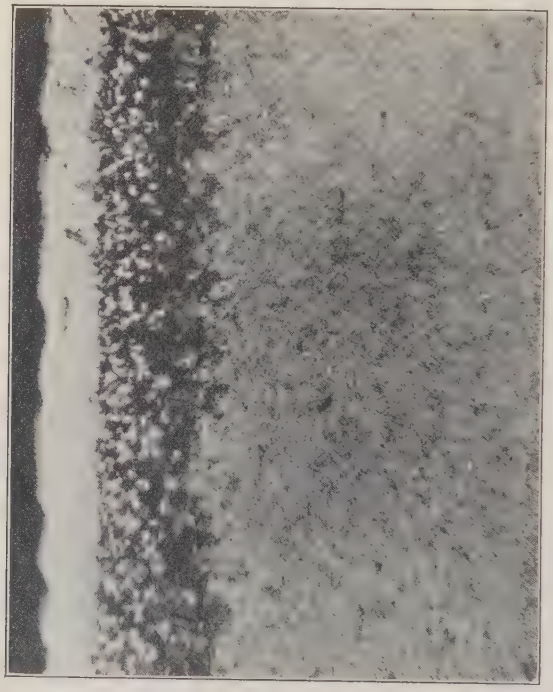


Fig. 2.—100 Diameters.

ture as ordinates, it is also observed that the decarbonization increases at a definite rate as the temperature is increased. (See Fig. 5.)

A similar experiment was then performed by annealing the test pieces for 3 hr. in a tube containing air and to which air had slight access through a small opening in one end. As shown in the following table, the results are very similar to those obtained in an atmosphere of CO_2, with the exception that the rate of decarbonization is now slightly less at all temperatures:

FIG. 3.—100 DIAMETERS.

FIG. 4.—100 DIAMETERS.

Piece No.	Temperature, ° F.	Depth of Decarbonization, Inches
1	1,200	0.000
2	1,300	0.000
3	1,400	0.004
4	1,500	0.005
5	1,600	0.014
6	1,700	0.016
7	1,800	0.025
8	1,900	0.039
9	2,000	0.064

The curve plotted from these results is shown in Fig. 5.

A similar experiment was then performed by annealing the pieces for 3 hr. in a current of pure, dry oxygen.

The results are very similar to those obtained in air and CO_2.

The curve plotted from the following table is shown in Fig. 5.

Piece No.	Temperature, ° F.	Depth of Decarbonization, Inches
1	1,300	0.000
2	1,400	0.002
3	1,500	0.006
4	1,600	0.012
5	1,700	0.017
6	1,800	0.027
7	1,900	0.040
8	2,000	0.068

A test piece was then annealed for 3 hr. at 2,000° F. in a current of pure, dry carbon monoxide. Microscopic examination revealed no trace of decarbonization. The CO used was first freed from oxygen and CO_2 by passing through a tube filled with highly heated carbon, and then through several wash bottles containing caustic potash solution. A flask of lime water was placed in the train at both the entrance and the exit of the furnace, to detect any traces of CO_2 either entering or escaping.

A second test piece was then annealed in the same manner, except that a clay boat filled with hammer scale (Fe_3O_4) was placed in the furnace in such a position that the current of carbon monoxide would pass over it before reaching the steel. As soon as the furnace reached its temperature it was observed that a gas was being evolved within the furnace. A noticeably larger quantity of gas was escaping from the furnace than was entering it. This would indicate that the Fe_3O_4 was probably giving up part of its oxygen. This evolution of gas continued through the entire period of annealing. The gases escaping from the furnace were

found to contain considerable quantities of CO_2. The CO_2 was probably formed by the following reactions:

$$Fe_3O_4 = 3FeO + O.$$
$$CO + O = CO_2.$$
$$FeO + CO = Fe + CO_2.$$

Microscopic examination of the annealed specimen revealed that the decarbonization had penetrated to a depth of 0.096 in.

To test the effect of water upon decarbonization, a test piece was annealed at 1,700° F. in a current of steam, the water from which the

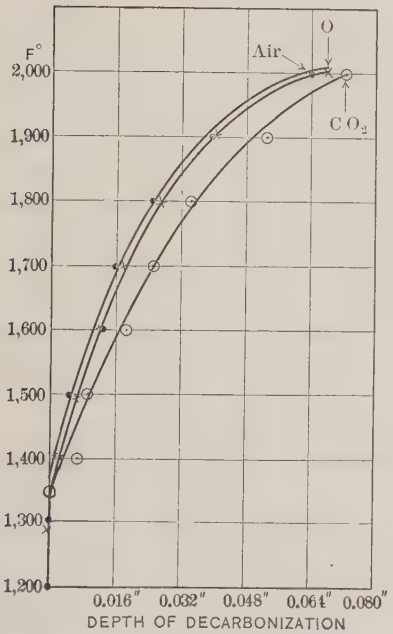


FIG. 5.—DECARBONIZATION OF STEEL ANNEALED IN AIR, OXYGEN, AND CARBON DIOXIDE.

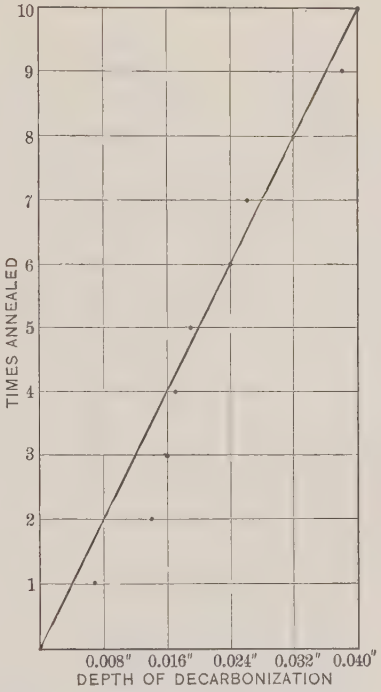


FIG. 6.—EFFECT OF REPEATED ANNEALING ON DECARBONIZATION OF STEEL.

steam was derived being first freed from CO_2 and air by prolonged boiling. Examination of the annealed specimen revealed a decarbonization to a depth of 0.030 in.

To test the effect of repeated annealings, a test piece was first packed in a tube with a small amount of charcoal, air being given limited access. It was then repeatedly annealed for 10 hr. each time, at a temperature of 1,450° F. The decarbonization was measured after each annealing, the

results being shown in the following table. The action was due to the CO_2 formed by the charcoal and the oxygen of the air.

No. of Times Annealed	Depth of Decarbonization, Inches
1	0.009
2	0.015
3	0.017
4	0.015
5	0.019
6	0.024
7	0.026
8	0.032
9	0.038
10	0.040

The curve plotted from these results shows that the rate of increase in depth of decarbonization with repeated annealings is a constant. (See Fig. 6.)

In all of the experiments referred to above, decarbonization has been effected by oxygen in the form of a gas. In order to determine if it is necessary for the oxygen to be in the form of gas to combine with the carbon in the steel, the following experiment was performed: A crucible of lead was saturated with lead oxide by the addition of litharge. A test piece was annealed in it for 3 hr. at 1,500° F. Microscopic examination revealed decarbonization at a depth of 0.024 in. This shows that easily reducible oxides such as Pb_2O, when dissolved in a molten bath, may also effect the elimination of carbon.

The conclusions to be drawn from these experiments are several:

It is shown that carbon is removed from the steel by a process of oxidation.

The oxidation is effected by either gases or liquids.

The principal active agents of the gases are air, CO_2, and water.

Of the liquids any good oxidizing agent dissolved in a liquid bath may be the active agent.

During the annealing the CO_2 necessary for the elimination of carbon may be formed through the reduction of iron oxide by CO.

No case has been observed where carbon was eliminated while the steel was heated below its critical point. This indicates that it is necessary for the carbon to be dissolved in the iron before decarbonization can take place.

It has been shown that the rate of decarbonization under uniform conditions is constant, without regard to the depth to which it has proceeded. From this we may infer that the carbon is transferred from the interior to the surface by diffusion through the solid solution. The oxidation takes place only at the surface.

Practical Applications

The decarbonization of steel is of great practical interest to every user of tool-steel products. With but few exceptions, all tool steel furnished by the makers shows decarbonization to a greater or less degree. This is the result of various heating processes during manufacture. The decarbonized surface, or "bark," as it is commonly called, must be removed before the steel is used for cutting tools, wearing surfaces, etc. The depth of bark may vary greatly owing to oxidizing conditions during the hammering, rolling, and annealing of the rough material.

The most common method of inspection consists of hardening a piece of the material, breaking, and examining the hardened fracture by means of the naked eye or hand magnifier. The bark is thus distinguished by its fracture being coarser than the high-carbon interior. To examine the reliability of this method the following experiment was performed:

A bar of $\frac{1}{2}$-in. round crucible steel was selected, having the same composition as the one used in the previous series of experiments. It was annealed and examined microscopically. The bark was found to be approximately 0.015 in. in depth. Pieces were cut from this bar and hardened at various temperatures. Microscopic examination was then made of a piece hardened at each temperature, the decarbonization being measured to the point where the normal structure of the interior began.

The depths of bark shown by the microscope on the various pieces were as follows:

Piece No.	Temperature of Hardening, ° F.	Depth of Bark, Inches
1	1,370	0.015
2	1,380	0.018
3	1,390	0.018
4	1,415	0.018
5	1,435	0.021
6	1,460	0.020
7	1,490	0.024
8	1,530	0.027

The increase in the amount of bark observed with the increase in temperature is of course due to the fact that an increased amount of cementite is held in solution by quenching from the higher temperatures.

The examination of the hardened fractures gave the following results.

No. 1 was only slightly hardened. The bark scarcely shows at all in the fracture.

No. 2 gives only a very slight indication of bark.

No. 3 no indication of bark.

No. 4 very slight indication of bark.

No. 5 bark shows more distinctly but only in spots.

No. 6 bark shows fairly distinctly.

No. 7 bark very distinct.

No. 8 the fracture of the interior has become so coarse that there is no distinction between it and the bark.

This method of inspection is therefore reliable only when the hardening heats are carefully regulated and the results frequently checked by microscopic examination.

The method of inspection which the author has found to be the most satisfactory is by microscopic examination of an annealed specimen. The amount of carbon present and the exact condition of the bark can be determined very readily. Measurements of the depth of decarbonization may be made with a micrometer eye piece.

In the machining of tool steel, bark presents serious difficulties. It is a well-known fact among machinists that tool steel is frequently very difficult to machine near the surface. Also that after this hard bark is removed very little difficulty is encountered. This condition is almost invariably due to the surface decarbonization of the steel.

A typical example of this difficulty is found in turning twist-drill blanks from the round bar. Here it is sometimes observed that the outside of the bar, to a depth of several thousandths of an inch, is very soft. Beneath this is a ring about $\frac{1}{32}$ in. in thickness, so hard that it cuts a nick in the lathe tool. Below this ring the piece is uniformly soft.

Microscopic examination of such a piece reveals that the extreme outside of the bar consists of either ferrite or ferrite mixed with pearlite, which would of course machine as easily as machinery steel. The hard ring is found to consist of nearly pure pearlite in very coarse crystalline grains. The soft interior consists usually of finely divided pearlite and globular cementite, which also machines with great ease. Such a bark is shown in Fig. 7. The structure of the band of pure pearlite magnified 1,000 diameters is shown in Fig. 8.

In case a piece of tool steel is being machined in such a way that only a part of this layer of coarse pearlite is removed still greater difficulty is encountered. The surface of the work will be rough and the point of the tool will be worn out with great rapidity. The rapid wearing of the tool makes it impossible to hold a uniform size on any number of pieces of work.

The cause of this layer of coarse crystals is the difference between the critical points of eutectoid and hypereutectoid steel. At the correct temperature for annealing the hypereutectoid steel, the eutectoid is far enough above its critical point to allow the crystalline grains to grow to considerable size.

The fact that eutectoid steel with the pearlite arranged in coarse

FIG. 7.—100 DIAMETERS.

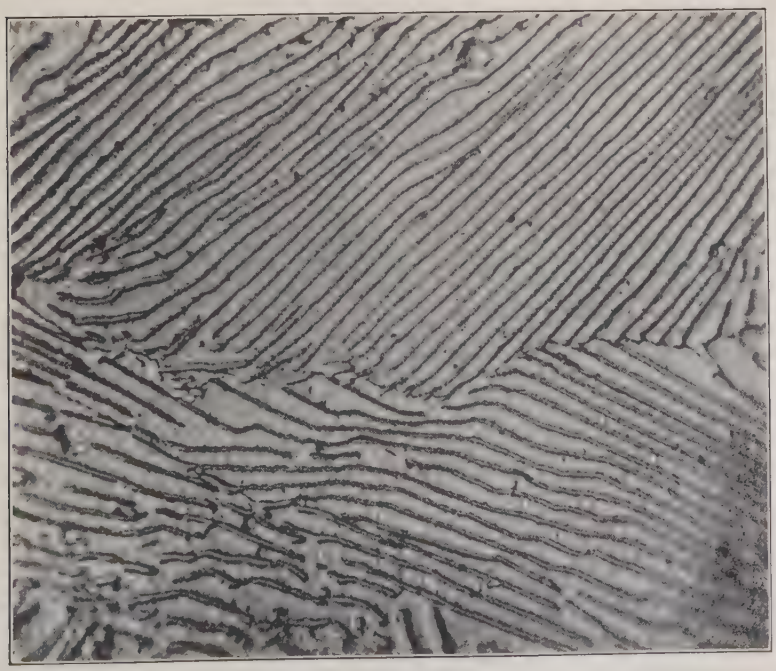


FIG. 8.—1,000 DIAMETERS.

grains is more difficult to machine than hypereutectoid steel in fine grains seems at first surprising. This is easily explained when we consider the manner in which the steel yields under stress and effect of the various micro-constituents upon the cutting tool.

It has been shown by several investigators that a tool when removing a chip from a piece of metal splits it off by an action similar to a wedge. In this case the hardest wear on the tool occurs not at the extreme point but a short distance back from the point where the pressure of the chip is greatest. Fig. 9 shows this condition on an exaggerated scale.

It will readily be seen that there is always a crack running ahead of the tool as it is forced through the material that is being cut. When the tool is cutting in coarse pearlite the crack will naturally follow the lines

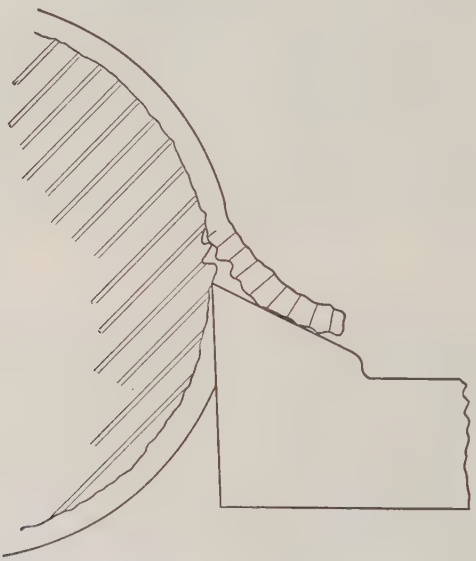


Fig. 9.—Diagram Illustrating Action of Metal-Cutting Tools.

of least resistance. These are the boundary lines of the grains and through the grain in the direction of the laminations. This being the case, it follows that the larger the size of the crystalline grains and the more distinct the lamellar structure, the more irregular will be the crack and the rougher the resulting surface. Such a crack passing through a mass of coarsely crystalline pearlite is shown in Fig. 10.

The very rapid wear on the cutting tool engaged in cutting coarse pearlite is probably due to two causes:

First, the rough surface left by the crack running ahead of the tool greatly increases the amount of work and friction on the extreme cutting edge. The point of the tool is obliged to rub over many protruding grains

FIG. 10.—500 DIAMETERS.

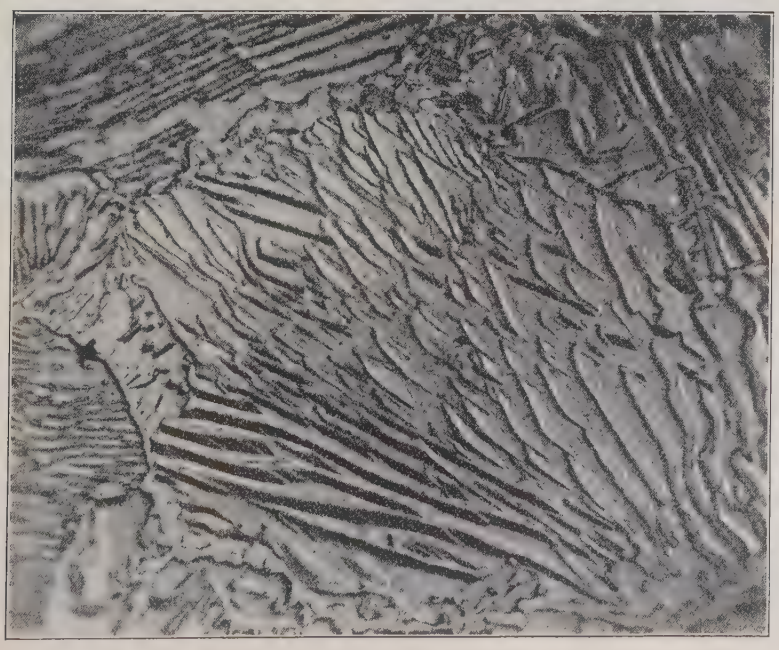


FIG. 11.—1,000 DIAMETERS.

of pearlite and sometimes to work in and tear others out entirely. The result of this extra work is to round off the cutting edge rapidly and increase the friction, until the point is entirely destroyed by the heat generated.

The second cause of the extreme wear on a cutting tool working in coarse pearlite is the great abrasive quality of cementite when firmly held by the alternate plates of ferrite. The sharp angular plates of hard cementite in coarsely lamellar pearlite have a much greater abrasive action than the rounded masses of globular cementite in well-annealed hypereutectoid steel. A grain of coarsely lamellar pearlite exhibiting this feature is shown in Fig. 11.

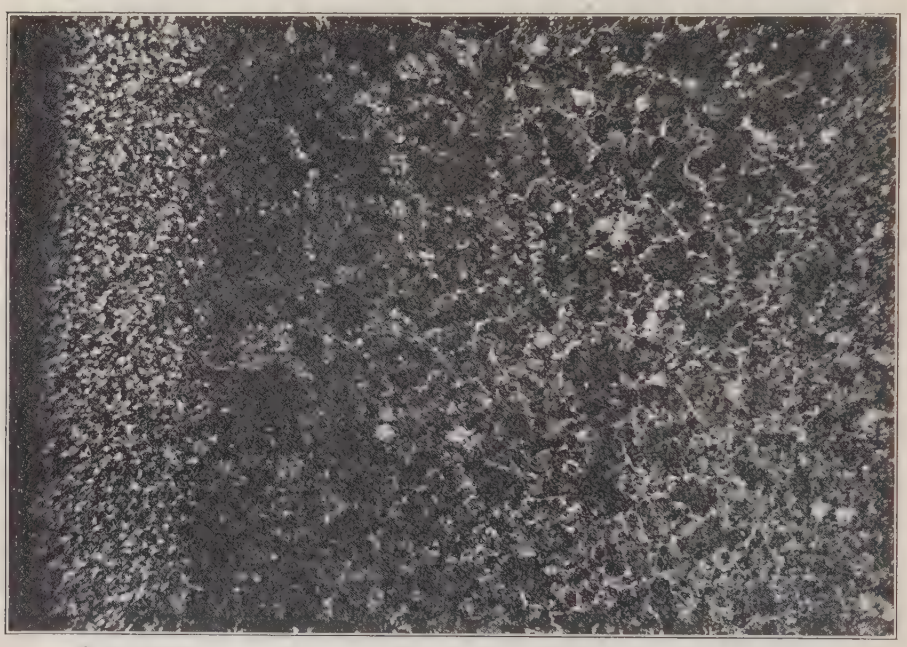


Fig. 12.—50 Diameters.

A still more difficult form of bark to machine is one which has a band of coarse hypereutectoid steel below the band of coarse pearlite. In this case the coarse grains of pearlite are bounded by a cementite network, which seems to add greatly to the difficulty of machining. Fig. 12 shows a bark in which this phase is marked. At the edge of the bar is a band about 0.017 in. in width of mixed ferrite and pearlite in small grains. Below this is a band of pure pearlite about 0.030 in. in width, with large crystalline grains. Below this is a band of coarse-grained hypereutectoid steel about 0.030 in. in width, the grains of pearlite being surrounded by a coarse network of cementite. Below this is the normal

structure of the interior, which is very fine-grained pearlite with small masses of globular cementite. In this piece of steel the band of ferrite mixed with pearlite and the hypereutectoid interior of the bar are very easy to machine. The bands of coarse pearlite and coarse pearlite surrounded by a cementite network are extremely difficult to machine.

If a piece of tool steel upon which a bark yet remains is hardened it is apparent that the surface will be hardened only in proportion to the carbon it contains. This may vary down to the point where the surface does not contain enough carbon to harden at all. In the great majority

FIG. 13.—100 DIAMETERS.

of cases where tool steel is used for bearings, dies, and cutting tools the most severe wear must be borne by the surface or extreme edge. For instance, in the case of twist drills, a bark remaining on the tool would cause the outside point of the cutting edge, where the speed and wear are greatest, to be soft. This would result in the rapid wearing of the outside corner, causing the drill to run undersize and so increase the friction as soon to destroy the whole point of the drill. The effect of lowering the carbon content of the surface of a die or bearing even 10 or 20 points is very apparent.

Bark is also a frequent cause of checks and firecracks in hardened tools. One illustration of this difficulty is shown in Fig. 13, where there

is a sharp line of demarcation between the bark and the structure of the interior. In this case, when the tool is quenched, a crack is liable to start and run along the boundary between the bark and the interior.

A second case in which bark left on a tool may start firecracks is where the coarse network of cementite, as shown in Fig. 12, is present. This cementite network is very liable to start a check or firecrack which will ruin the tool. The manner in which a cementite network starts hardening cracks has been more fully discussed in a previous paper.[4]

These remarks on the practical applications of a knowledge of the decarbonization of tool steel include but a few of the many difficulties which may be traced to this source. Each manufacturer and user of tool-steel products has of course his own difficulties with decarbonization, which can be solved only by research within his own shop.

In view of the effect of decarbonization upon the quality of tool-steel products it is surprising that so small an amount of attention has been paid it.

Discussion

Albert Sauveur, Cambridge, Mass.—Mr. Emmons is to be congratulated for having placed before us in so forcible and satisfactory a manner a question of much importance—that of the surface decarbonization of steel. Of course, both manufacturers and consumers have known for many years that during the heat treatment of high-carbon steel, surface decarbonization is always likely to occur to a greater or less degree, but the subject does not seem to have been investigated with the method and thoroughness it deserves.

I am in substantial agreement with Mr. Emmons's conclusions, and I have but little to add to what he has so well said. His statement that carbon dioxide is just as effective an oxidizing agent as pure oxygen is likely to cause surprise and some will wonder whether his investigation has been carried far enough to warrant so unexpected a conclusion. The reaction $C + O = CO$, expressing the oxidation of carbon by pure oxygen, is decidedly exothermic, whereas the reaction $C + CO_2 = 2CO$, expressing the oxidation of carbon by carbon dioxide, is endothermic. Decarburization by pure oxygen, therefore, should take place much more readily than by carbon dioxide.

One cannot help regretting that the author does not mention the best way of preventing in a practical manner surface decarbonization while treating tool steel and other high-carbon steels.

Bradley Stoughton, New York, N. Y.—In connection with the very interesting paper which Mr. Emmons has presented, reference should be had to a paper[5] bearing in an interesting way on the results that Mr. Emmons obtained in decarbonization, which was prepared by

[4] *Iron Age*, vol. lxxxix, No. 8, p. 472 (Feb. 22, 1912).

[5] The Decarburisation of Steels in the Salt Baths used for Heating Prior to Hardening.

A. M. Portevin for presentation at the meeting of the Iron and Steel Institute of London in September, 1914.

E. G. Spilsbury, New York, N. Y.—The point brought out by Mr. Emmons regarding the activity of carbon dioxide as an oxidizing agent, has been fully recognized in the manufacture and distillation of zinc from its ores. In the presence of carbon dioxide, metal in its nascent condition is very rapidly decomposed.

One of the great difficulties in the distillation of zinc arises from the great affinity of the metallic vapor for oxygen, and the aim in the whole process is to exclude the presence of oxygen as much as is possible. This is why we have not been able up to the present time to improve on the use of comparatively small units of distilling retorts.

Even then, from the process itself, carbon dioxide in considerable quantities is formed, and at the temperature inside the retort would very actively react on the metal vapor. It is therefore necessary to pass the vapor as rapidly as formed into a zone of temperature below the point at which the carbon dioxide becomes inert. In this case the reduction of temperature from the point of volatilization of 1,100° to 700° is necessary. This may have some bearing on the question Mr. Emmons brings up.

Oliver W. Storey, Pittsburgh, Pa. (communication to the Secretary*).—While carrying on researches in the annealing process for malleable castings the writer became interested in the surface effects produced during the annealing of white iron. It was noticed that various commercial malleable products differed, not in the constitution of the "black heart," but in the surface layers. Some products showed a skin consisting entirely of pearlite; others a skin of two layers, the outer of ferrite, beneath which was one of pearlite; while some had a skin consisting entirely of ferrite.

In the investigation a large number of packing materials were used for the annealing process. These included oxidizing materials, neutrals, and materials evolving carbon monoxide.

The results showed that the chemical nature of the packing material was usually of little importance, with a few exceptions. The method of packing the castings was of greatest importance. This influenced the circulation of the furnace gases in the annealing boxes, since closely packed boxes allowed less circulation than those loosely packed. Therefore, the composition of the gases in the annealing furnaces was also important. It was found that in those boxes which were loosely packed, readily allowing circulation of air (an electric ribbon-wound furnace being used as the heating medium), the resulting malleable iron had a skin which consisted almost entirely of pure ferrite, with little pearlite, due to the oxidation of the surface carbon by the air. Where the packing material consisted of a closely packed mass such as finely ground fire clay

* Received Oct. 15, 1914.

the skin consisted of ferrite and pearlite, since the oxidation was retarded. Where ground carbon was used the surface usually consisted of pearlite, though here the temperature of annealing was important.

When a highly oxidizing packing was used all of the carbon was removed from the surface. If a case-carbonizing mixture at a high annealing heat (900° C.) was used the surface of the malleable iron consisted of a high-carbon steel, giving a hard surface. When carbon was used as a packing material the temperature was important owing to the equilibrium conditions of the reversible reaction: $CO_2 + C \rightleftarrows 2CO$.

At the lower temperatures CO_2 is in excess and castings annealed at these temperatures had a decarbonized surface though packed in carbon. This was due to the action of CO_2, as pointed out by Mr. Emmons in his paper. At the higher temperatures more CO was present and the surface consisted of pearlite.

From the results it will be seen that in the malleable process the composition of the furnace gases and the method of packing influence the skin structure.

In a paper before the September meeting of the American Foundrymen's Association the writer called attention to the surface action of both CO and CO_2 and the commercial advantages in being able to obtain a soft carbonless rim or one of high-carbon steel of excellent wearing qualities. The results obtained in the researches of Mr. Emmons, with slightly modified conditions, are directly applicable to the making of malleable iron.

H. O. Hofman, Boston, Mass. (communication to the Secretary*).—The explanation for the surface decarbonization of tool steel by J. V. Emmons is based in part on the experiments of F. Wüst (*Metallurgie*, vol. v, No. 2 (1908)) which show that *in an evacuated tube* ferric oxide gives off some of its oxygen at 600° C. and continues to do this as the temperature is raised. Thus, at

1,050° C., Fe_2O_3 is changed into $Fe_2O_3.FeO$ (= Fe_3O_4).
1,100° C., Fe_3O_4 is changed into $Fe_2O_3.2FeO$ (= Fe_4O_5).
1,200° C., Fe_4O_5 is changed into $Fe_2O_3.3FeO$ (= Fe_5O_6).

It has been proved experimentally by three investigations that *at atmospheric pressure* ferric oxide is not decomposed by heat alone at 1,300° C.

K. Honda and T. Sone (*Science Reports of the Tohoku Imperial University*, Sendai, Japan, vol. iii, p. 223 (1914)) found that hematite remained unchanged when heated to 1,300° C., in a current of air.

W. Mostowitsch and H. O. Hofman (*Trans.*, xl, 807 (1909)) at an earlier date heated ferric oxide in a current of air to 1,500° C. and found no change in weight.

E. G. Kohlmeyer (*Metall und Erz*, vol. x, pp. 447 to 483 (1913)) found that in a current of oxygen at 920° to 950° C. the red Fe_2O_3 became black,

* Received Dec. 21, 1914.

lost oxygen as the temperature was raised, and fritted at 1,370° with the formation of $Fe_2O_3.FeO$.

The new compound gave up more oxygen with increase of temperature, fused at 1,470°, and formed $5Fe_2O_3.3FeO$; this further lost oxygen up to 1,520°, when there was formed $4Fe_2O_3.3Fe_2O_3$; and the last gave up oxygen as the temperature was raised, and fused at 1,600° with the formation of $Fe_2O_3.FeO$, which corresponds to the well-known Fe_3O_4.

J. V. Emmons (communication to the Secretary*).—The writer is deeply indebted to Professor Sauveur for the favorable comments on his paper. The fact that carbon dioxide is as powerful an oxidizing agent as pure oxygen was at first surprising, although it is well known that at temperatures above the critical points of high-carbon steel it is slightly dissociated into CO + O. It was shown by the several experiments with different oxidizing agents that the rate of decarbonization was similar in all cases, and that in the presence of any oxidizing agent the rate of decarbonization was a function of the temperature only. This was taken as an indication that the rate of decarbonization was dependent upon the rate of diffusion of Fe_3C through gamma iron. Further confirmation of the activity of carbon dioxide as an oxidizing agent at high temperatures is found in the discussion by Mr. Spilsbury and Mr. Storey.

The prevention of decarbonization is obviously a question of preventing the access of oxidizing agents to the steel while heated above its critical points. The writer regrets that no simple rule for doing this can be laid down. The heat treatment of tool steel assumes such a variety of forms that the accomplishment of this end is largely a matter of ingenuity in dealing with specific problems.

The paper of Mr. Portevin is especially interesting for the information given in regard to the equilibrium between carbonization and decarbonization. His results indicate in a striking manner that the rate of carbonization as well as decarbonization is a function of the rate of diffusion of the carbide through the solid solution of Fe_3C in gamma iron.

There is no question that a knowledge of the mechanism of decarbonization is of great value in the production of malleable iron, as Mr. Storey so well says. The commercial possibilities of a skillful regulation of the condition of the carbon in the skin or rim of the casting are very great.

The writer will not dispute the contention of Mr. Hofman that ferric oxide is not decomposed by heat alone at atmospheric pressure, as in all the experiments performed there was present carbon monoxide, which would serve to start the decomposition. Indeed, in the commercial heat treatment of tool steel carbon monoxide is almost invariably present. Further confirmation of this decomposition of ferric oxide will be found in the work of C. M. Johnson. (*Journal of Industrial and Engineering Chemistry*, vol. i, No. 7, p. 459 (July, 1909)).

*Received Mar. 10, 1915.

The Heat Treatment of Steel Castings

BY C. D. YOUNG, O. D. A. PEASE, AND C. H. STRAND, ALTOONA, PA.

(New York Meeting, February, 1914)

In an effort to employ cast steel of a stronger structure than that found in the annealed steel castings, the possibilities of heat treatment which will increase the strength without materially decreasing the ductility may be resorted to.

The following abstracted report (which is a brief outline of what has already been done by the Pennsylvania Railroad Co. at its shops and Test Laboratories at Altoona, Pa.) is of material interest, as it indicates what may be done with steel castings when properly treated, thereby permitting in railway service greater strength of cast steel parts without any increase in weight or space.

The question of the heat treatment of alloy steel castings is not taken up in this paper.

The obscurity formerly surrounding the heat treatment of steel has been for the most part removed by the development of our knowledge of the critical points of steel, pyrometers, furnace construction, and the testing of the finished product.

The operations of the heat treatment proper will be taken up under the heads of (1) heating for quenching; (2) quenching; (3) drawing.

(1) *Heating for Quenching.*—Heating for quenching is best conducted slowly, especially in the case of castings of variable thickness. Cracks may occur either in heating or in cooling, due to different temperatures at different points of the casting. The castings should be thoroughly soaked at the maximum temperature (generally 1,500° to 1,600° F.), 1 hr. being sufficient for sections 1 ft. in thickness. The minimum temperature which will produce the desired hardening effect will, in all cases, be found to be the most satisfactory, as the grain coarsens when the critical range is exceeded to too great an extent. All temperatures should be governed by a checked pyrometer with the hot junction to the heated object, and with several couples in a large furnace to insure a uniform temperature.

(2) *The Quenching Operation.*—The casting should be transferred as quickly as possible from the furnace to the quenching bath, and in the case of large castings, such as locomotive frames, this is by no means a

simple matter. The larger castings are best handled by means of cranes and rollers.

The quenching agent employed is generally water or oil, preferably the former, because of its cheapness and drastic cooling effect, more readily breaking up the coarse cast-steel grain. With intricate castings it is generally best to use oil. With water, it is possible to have a large tank and a large running stream, serving to maintain a uniform temperature. Castings should never be thrown in to rest on the bottom of the tank, but should be agitated to prevent the formation of a coating of vapor, retarding the quenching effect. It is also best, whenever possible, to quench the thicker portions first.

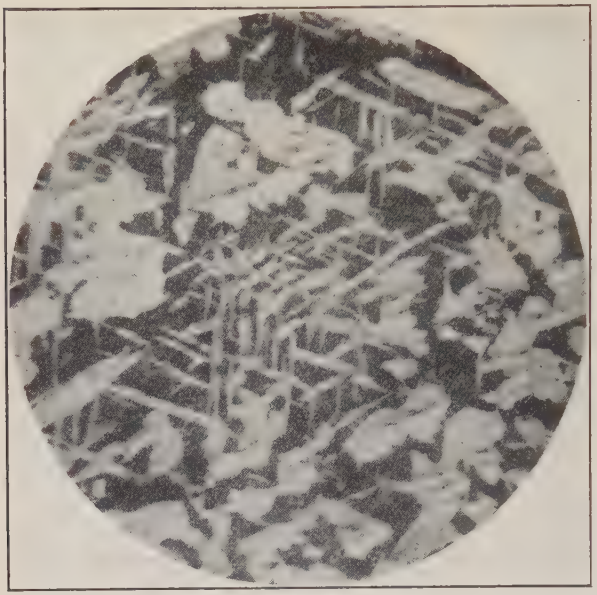
(3) *The Drawing Operation.*—Whenever possible, the drawing should be done in a bath of some kind, such as lead, barium chloride, a barium chloride-salt mixture, or oil. In the case of large castings this is manifestly impossible, and great care should be exercised in obtaining a uniform temperature in the drawing furnace.

The use to which the casting is to be put determines the drawing temperature, railroad work, by reason of the shock and vibration of the road, requiring high ductility at the sacrifice of some strength.

The results of some tensile, chemical, and micrographic tests of commercially annealed and experimentally heat-treated cast-steel bolsters accompany this paper. The following table shows the heat treatment and the results of tensile and chemical tests:

	Commercially annealed				Heat treated			
Bolster No.	B43	B18	B2	B29	B10	B39	B15	BB
Elastic limit, pounds	30,950	36,700	36,070	41,990	44,663	61,940	61,390	54,930
Ultimate strength, pounds	57,610	78,665	74,020	69,783	80,393	87,890	92,750	84,290
Elongation in 2 in., per cent	14.8	6.0	12.3	5.3	13.2	4.3	11.7	17.3
Reduction of area, per cent	26.8	4.5	16.0	5.9	19.0	7.7	15.4	27.3
Analysis								
Carbon, per cent	0.31	0.51	0.35	0.49	0.32	0.33	0.33	0.30
Manganese, per cent	0.82	0.68	0.63	0.70	0.62	0.88	0.76	0.62
Silicon, per cent	0.34	0.31	0.36	0.34	0.26	0.41	0.37	0.36
Phosphorus, per cent	0.016	0.045	0.036	0.036	0.042	0.014	0.041	0.032
Sulphur, per cent	0.010	0.030	0.016	0.042	0.030	0.025	0.029	0.018
Heat treatment								
Annealed at, F.°	1,500	1,550	1,550	1,550	1,580	1,640	1,600	.....
Quenched in water at, F.°	.....	.....	.....	.....	1,600	1,600	1,600	1,600
Drawn to, F.°	.....	.....	.....	.....	900	900	900	900

Examination of the accompanying photomicrographs, Figs. 1 to 8, inclusive, shows the inefficiency of the manufacturer's annealing, which is by no means uncommon. The contrast between the annealed samples and the treated samples is readily apparent. Figs. 1 and 3 show a coarse needlelike ferrite formation, traces of the casting structure, not obliterated by annealing. Figs. 2 and 4 show the structure of some



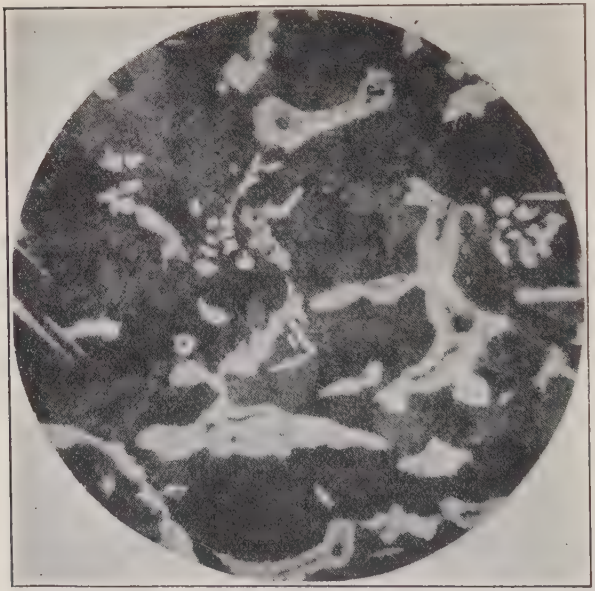
Magnification 100 diameters. Etched with picric acid.

FIG. 1.—No. B43. COARSE NEEDLE-LIKE FERRITE STRUCTURE, INDICATING POOR ANNEALING.

CAST-STEEL BOLSTERS, COMMERCIALLY ANNEALED

Physical Test

	B43	B18
Ult. str., lb. per sq. in.	57,610	78,665
Elas. lim., lb. per sq. in.	30,950	36,700
Red. of area, per cent.	26.8	4.5
Elong. in 2 in., per cent.	14.8	6.0



Magnification 100 diameters. Etched with picric acid.

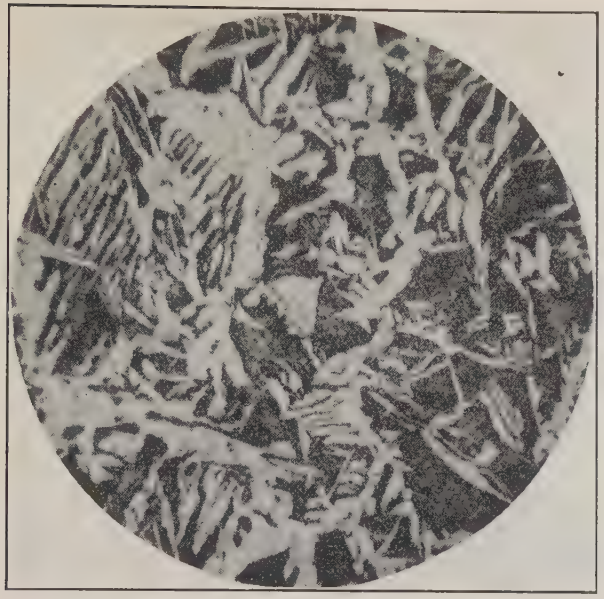
FIG. 2.—No. B18. COARSE HIGH-CARBON STRUCTURE. THE FERRITE IS SEGREGATED IN LARGE AREAS.

Chemical Analysis

	B43	B16
Carbon	0.31	0.51
Manganese	0.82	0.68
Silicon	0.34	0.31
Phosphorus	0.016	0.045
Sulphur	0.010	0.030

Heat Treatment

Annealed at..... 1,500° F.



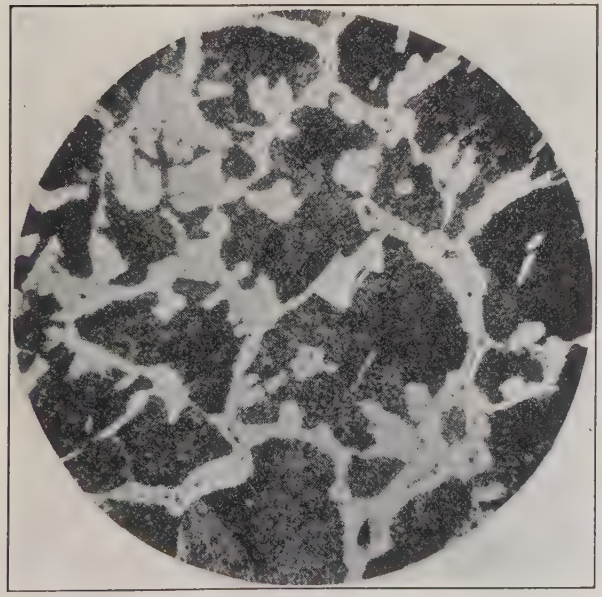
Magnification 100 diameters. Etched with picric acid.

FIG. 3.—No. B2. COARSE NEEDLE-LIKE FERRITE STRUCTURE, INDICATING POOR ANNEALING.

CAST-STEEL BOLSTERS, COMMERCIALLY ANNEALED

Physical Test

	B2	B29
Ult str., lb. per sq. in.	74,020	69,783
Elas. lim., lb. per sq. in.	46,070	41,990
Red. of area, per cent.	16.0	5.9
Elong. in 2 in., per cent.	12.3	5.3



Magnification 100 diameters. Etched with picric acid.

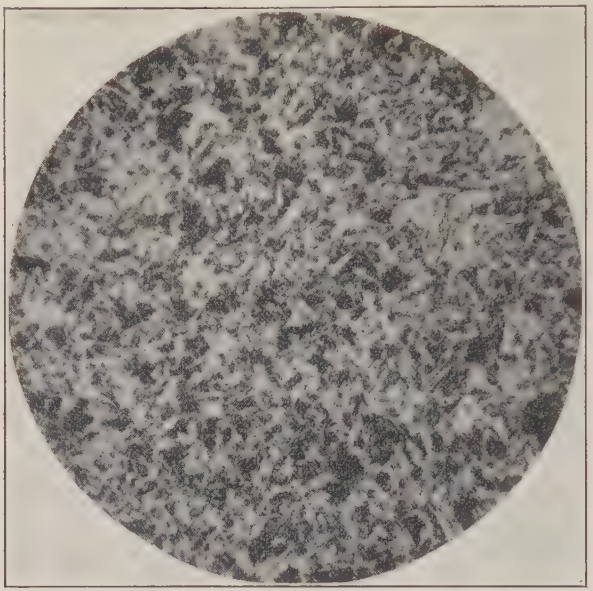
FIG. 4.—No. B29. COARSE HIGH-CARBON STRUCTURE. THE FERRITE FORMS A THICK NETWORK.

Chemical Analysis

	B2	B29
Carbon	0.35	0.49
Manganese	0.63	0.70
Silicon	0.36	0.34
Phosphorus	0.036	0.036
Sulphur	0.016	0.042

Heat Treatment

Annealed at ..... 1,550° F.



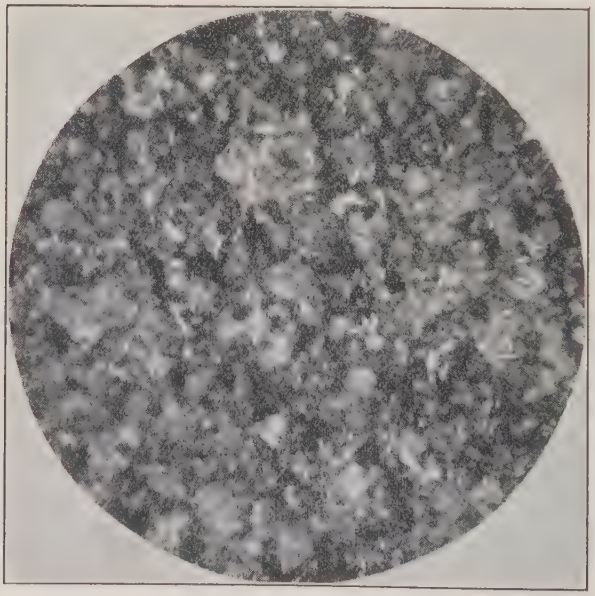
Magnification 100 diameters. Etched with picric acid.

FIG. 5.—No. B10. GOOD HEAT-TREATED STRUCTURE. THE LARGE AMOUNT OF FERRITE LENDS DUCTILITY.

CAST-STEEL BOLSTERS, HEAT TREATED

Physical Test

	B10	B39
Ult. str., lb. per sq. in.	80,393	87,890
Elas. lim., lb. per sq. in.	44,663	61,940
Red. of area, per cent.	19.0	7.7
Elong. in 2 in., per cent.	13.2	4.3



Magnification 100 diameters. Etched with picric acid.

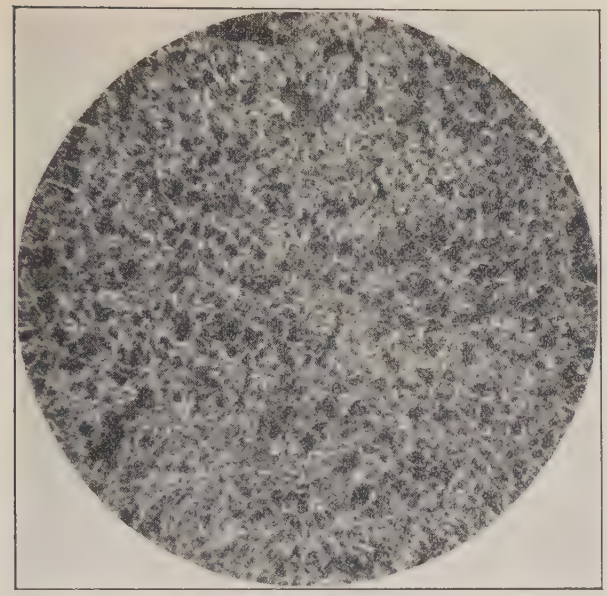
FIG. 6.—No. B39. SHOWS TOO LITTLE FERRITE, GIVING LOW ELONGATION.

Chemical Analysis

	B10	B39
Carbon	0.32	0.33
Manganese	0.62	0.88
Silicon	0.26	0.41
Phosphorus	0.042	0.014
Sulphur	0.030	0.025

Heat Treatment

	B10	B39
Annealed at	1,580	1,640
Quenched at	1,600	1,500
Drawn to	900	900



Magnification 100 diameters. Etched with picric acid.

FIG. 7.—No. B15. GOOD HEAT-TREATED STRUCTURE, SHOWING SMALL FINE FERRITE NETWORK.

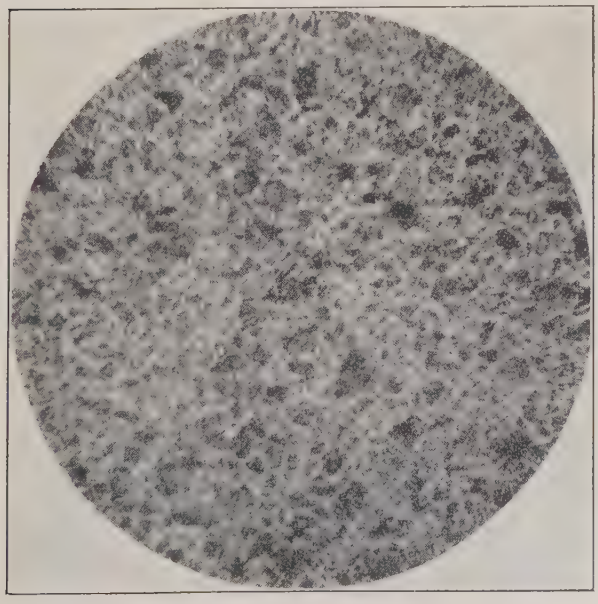
CAST-STEEL BOLSTERS, HEAT TREATED

Physical Test

	B15	BB
Ult. str., lb. per sq. in.	92,750	84,290
Elas. lim., lb. per sq. in.	61,390	54,930
Red. of area, per cent.	15.4	27.3
Elong. in 2 in., per cent.	11.7	17.3

Chemical Analysis

	B15	BB
Carbon	0.33	0.30
Manganese .	0.76	0.62
Silicon	0.37	0.36
Phosphorus .	0.041	0.032
Sulphur	0.029	0.018



Magnification 100 diameters. Etched with picric acid.

FIG. 8.—No. BB. GOOD HEAT-TREATED STRUCTURE, SHOWING SMALL FINE FERRITE NETWORK.

experimental high-carbon bolsters, which are not sufficiently ductile. Figs. 5, 7, and 8 show excellent heat-treated structures, with a very fine ferrite network. Fig. 6, while given the same heat treatment as the others, shows very little free ferrite, checked by the low elongation of the tensile test. This may be due to the high manganese and silicon content. The heat treatment of these castings would have been more satisfactory with a drawing temperature of 1,100° F. instead of 900° F.

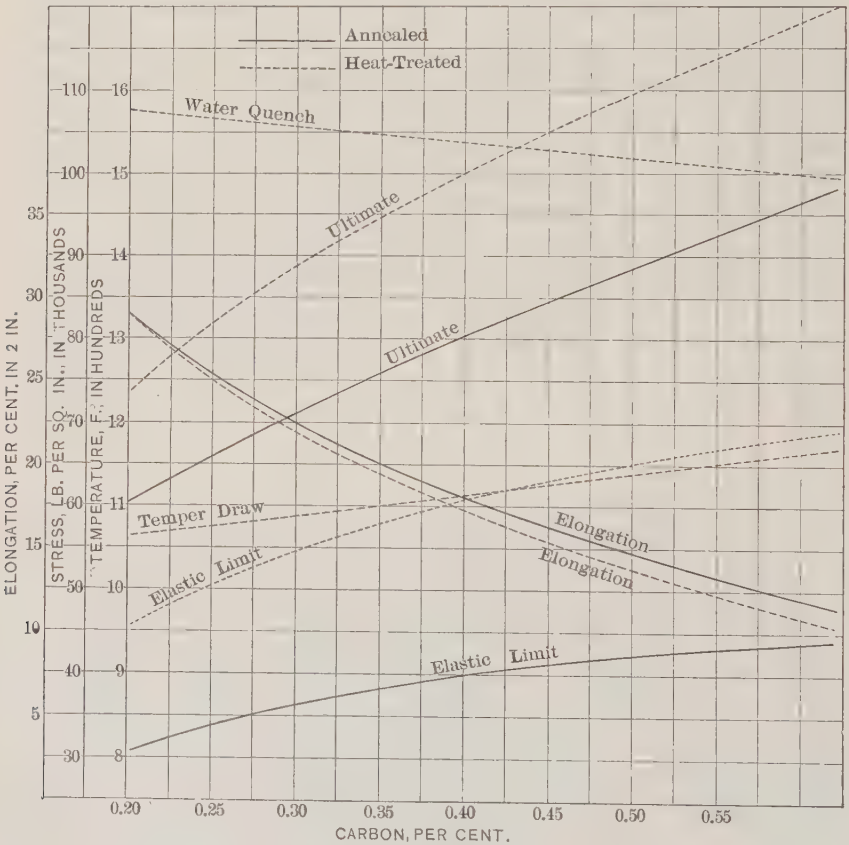


Fig. 9.—Average Results of Tensile Tests.

The plot, Fig. 9, shows the average results of a considerable number of tensile tests made from large castings. It will be observed that the heat treatment increases the elastic limit about 50 per cent. and the ultimate strength about 25 per cent., without any material change in elongation. The heat treatment to produce these results is also shown in the plot. In the event a fair amount of ductility is required, it is not desirable to have the carbon content above 0 35 per cent. Individual results will vary from 5 to 10 per cent. from the average results.

Discussion

C. H. Strand.—I would like to supplement the description of the photomicrographs by an interpretation of them by some figures which I have made which will perhaps be rather a departure from what has been done in the past.

Since this paper was written, there has been considerable agitation concerning the question of grain size in steel. As a result, Mr. Young, Engineer of Tests, P. R. R., has asked if it is not possible to devise a metallographic specification of necessarily wide tolerances. The results of my work in this connection have convinced me that the size and amount of the ferrite areas is the shortest and best means of judging the grain size in annealed and heat-treated steels. Ferrite has a tensile strength of 50,000 lb. per square inch; pearlite, 110,000 lb.; and sorbite, somewhat higher. Obviously, as the percentage of ferrite and size of individual ferrite areas decrease, the strength of the steel increases. However, there must be a maximum and minimum limit for this condition. In a steel with large ferrite areas fracture easily originates and progresses. Finding the method of measuring areas too tedious, I decided simply to measure the length of the longest axis or dimension of several of the largest grains in the field, average them, and calculate the theoretical number of these in a row per inch. I have estimated the grain size of the steel from the eight photomicrographs from this paper, which are as follows:

	Ferrite Cells per Inch
Annealed:	
Fig. 1, No. B43	14,000
Fig. 2, No. B18	13,000
Fig. 3, No. B2	15,000
Fig. 4, No. B29	27,000
Average, annealed	17,200
Heat Treated:	
Fig. 5, No. B10	52,000
Fig. 6, No. B39	100,000 ?
Fig. 7, No. B15	100,000
Fig. 8, No. BB	60,000
Average, heat treated	78,000

The average of annealed samples is 17,200 and of heat treated is 78,000, showing, as is to be expected, a much finer grain in the heat-treated steels. The results of B39 are not reliable as the amount of ferrite is small and indistinguishable.

In railroad work, the sections are for the most part large, few alloy steels are used, and the carbon content is fairly low, resulting in the presence of only the constituents ferrite, pearlite, sorbite, with some

troostite. I believe it possible to devise some roughly accurate short and very useful metallographic specification for annealed and heat-treated steel. I would be pleased to have opinions from those present on this matter and to exchange specimens and views on methods of carrying on this work, with the object of the adoption of a standard metallographic specification.

Edwin F. Cone, New York, N. Y.—I want to call particular attention to the results which are given here of commercially annealed steel. I have had several years' experience in the past in producing steel castings for railroad work, especially locomotive frames, and the point to be made at the outset, it seems to me, is that everything depends, in the case of large steel castings in particular, upon the location of the test bar or the piece of steel from which the tests are to be made. In the case of a bolster, the conditions under which these are annealed differ from the test bar that might occur on a locomotive engine frame. Also, the original microstructure is totally different in each case, rendering comparisons unfair.

I have never seen results in commercially annealed steel which will compare as to inferiority with the results given in the paper. Here is a tensile strength of 78,000 lb. per square inch with an elongation in 2 in. of only 6 per cent. Here is a tensile strength of 57,000 lb. per square inch, with carbon 0.31 per cent., with an elongation of only 14.8 per cent. Ordinarily, in ninety-nine cases out of a hundred, cast steel with a carbon of 0.31 will give a much higher tensile strength, and an elongation at least twice 14.8; or even two and a half times that. The 0.49 carbon is given here with a tensile strength of only 69,783 lb. per square inch, which is absolutely contrary to the results obtained from commercially annealed castings. Of course, a great deal depends on the location from which these tests were taken.

The heat-treated tests are very interesting, and especially the method of heat treatment. I have known for some time that experiments were being made, and have been very anxious to hear of the results. I feel somewhat surprised that the elongation and the reduction of area are so poor as compared with the tensile strength, and also that the elastic ratio is not higher than is shown here; that is, I mean by the elastic ratio the quotient derived by dividing the tensile strength into the elastic limit.

I want to add, particularly, that the same results and much better ones have been obtained experimentally, and also commercially, by what is called air tempering; that is, heating the castings to just about the recalescence point, and pulling them out in the air to cool. By this method one secures a higher elastic limit, as compared with the tensile strength and elongation, far superior to those shown in this table.

I would like to ask the authors from what location in the castings these

tests were taken, and whether the heat-treated tests as compared with the commercially annealed tests were from the same class of steel castings, locomotive frames, or whether they were taken from other pieces.

JOHN H. HALL, New York.—Mr. Young's most interesting and valuable paper is exceedingly welcome at this time, and to me at least has a strong personal interest. Some five years ago, when I was engaged in seeking to produce a cast steel that should be capable of resisting severe and suddenly applied stresses in service, I advocated the use of a steel of about 0.25 per cent. carbon, quenched in water or oil, and reheated. My superiors in the company by which I was then employed were startled, to put it mildly, by my proposal to subject ordinary steel castings to so drastic a treatment as quenching in cold water. I was fortunate enough, however, to be in the employ of men who could be "shown," and the treatment was adopted and has been used ever since.

Last June I presented a paper on the Heat Treatment of Cast Steel before the American Society for Testing Materials, and endeavored to set forth the advantages of heat treatment, properly so called, for cast steel. It was politely pointed out to me in discussion that though, of course, water quenching might be all very well to give fine microstructures to laboratory test bars, yet such treatment of real castings would be out of the question, on account of the danger of setting up serious stresses and cracking the castings.

My paper presented before this Institute at the last October meeting, on Shock Tests of Cast Steel, was read largely to show the advantages of heat treatment for cast steel. In the discussion of the paper, no criticism was offered of the methods of treatments suggested.

It is most gratifying, therefore, to find at this time that other shops have taken up the heat treatment of castings, and especially to read in Mr. Young's paper (p. 425) "The quenching agent employed is generally water or oil, preferably the former, because of its cheapness and drastic cooling effect, more readily breaking up the coarse cast-steel grain. With intricate castings it is generally best to use oil." Evidently others have found that the danger of cracking castings by quenching them in water has been greatly exaggerated, as I found five years ago; and that the fine microstructure and accompanying strength and toughness produced by quenching are advantages well worth securing.

Mr. Young states that 1 hr. at about 816° to 871° C. is sufficient for the first heating. I myself am inclined to consider this a little too short a heating, when the very best results are to be attained. Again, he gives temperatures for the second heating varying from 482° to 593° C., for a 0.30-carbon steel. It is not stated how long the castings were subjected to this temperature, and I should like to ask Mr. Young whether he has investigated the influence of the time of second heating upon the physical properties of the steel. My own experience has been that several

hours are needed to remove the quenching stresses and draw the hardness to the desired degree.

The reheating temperatures given seem to me rather low. Personally, I have preferred to reheat the castings for several hours to a temperature closely approaching Ar1, some 660° to 680° C., in order to obtain the greatest possible toughness, without sacrificing strength to too great an extent. Mr. Young presents no figures for shock toughness of his steels, nor, in the plot at least, does he mention contraction of area—a most important property. I am inclined to think that with such low drawing temperatures as Mr. Young gives in his paper, a shock test would show that the quenching hardness had not been sufficiently drawn. I am not quite clear on this point, however, in that Mr. Young's paper gives but four tests of 0.30-carbon steel, with an average elongation of but 11.6 per cent., and an average contraction of area of 17.3 per cent., while in the plot he gives an average elongation for 0.30-carbon steel, heat treated, of 25 per cent., and does not state the contraction of area.

The following 17 tests (Table I) were made upon coupons attached by their long sides to castings of an average weight of about 1,500 lb. These results were selected at random from some 2,000 tests. These castings averaged from 0.20 to 0.30 per cent. carbon, from 1.10 to 1.30 per cent. manganese, and about 0.30 per cent. silicon. Phosphorus and sulphur were about 0.05 per cent. They were heated 3 or 4 hr. at 900° C., quenched in water, and reheated for from 5 to 8 hr. to from 650° to 680°

TABLE I

Tensile Strength	Elastic Limit	Extension, Per Cent. in 2 in.	Contraction, Per Cent.	Frémont Kgm.	Bend ½ by 1 in. on 1-in. Mandrel
88,850	63,500	26.60	52.90	19.0	180
88,300	61,240	28.11	54.75	14.0	180
89,600	60,400	26.19	47.30	15.0	180
85,950	60,400	28.36	59.05	22.0	180
89,200	61,020	26.19	51.00	11.0	180
100,000	70,820	23.80	49.65	23.5	180
92,850	62,300	24.66	55.60	15.0	180
90,100	60,200	26.95	58.75	13.0	180
98,800	66,680	22.80	48.75	24.0	145
88,550	60,600	26.60	54.40	11.0	180
96,150	65,070	25.57	57.15	15.0	180
90,000	55,100	31.05	56.30	15.0	180
85,100	55,970	29.14	58.80	30.0	180
80,800	51,510	29.14	62.50	28.5	180
90,700	62,780	27.20	56.60	25.5	180
85,950	51,040	25.26	58.60	30.0	180
83,450	51,700	29.69	60.45	27.5	180

TABLE II

Source	Tensile Strength	Elastic Limit	Elongation in 2 in., Per Cent.	Contraction of Area, Per Cent.	Frémont Kgm.	Bend ½ by 1 in. on 1-in. Mandrel.
J. H. H., Table I.						
Average	83,780	60,020	26.9	55.44	19.9	177.6
Lowest	80,800	51,040	22.8	47.3	11.0	145.
Highest	100,000	70,820	31.05	62.5	30.0	180.
Young, Tests						
Average	86,330	55,730	11.6	17.3	.....	.....
Lowest	80,393	44,663	4.3	7.7	.....	.....
Highest	92,750	61,940	17.3	27.3	.....	.....
Young, Plot						
0.30 C. steel	89,000	55,000	24.0	.....	.....	.....

C. The very high tensile strength and elastic limit, coupled with high elongation, very high contraction of area, a nearly perfect bending test, and an excellent shock toughness, are very noticeable. In Table II, I have summarized Mr. Young's results and my own, in order to show that by the use of higher drawing temperatures (and a high percentage of manganese) it is possible to secure castings as strong as those of Mr. Young, but much tougher than those he shows in full. It would be most interesting, as I said before, if Mr. Young would supplement his excellent paper by giving us more detailed information on the tests upon which his plot is based.

O. D. A. PEASE.—Relative to the test specimens, as brought out earlier in the discussion, these test specimens were cut from the bolster itself, from that part of the bolster which was not strained during the time of the transverse test of the full-size bolster, and were taken outside of the supporting knife edges. It is interesting to note in connection with the manufacture of steel castings, such as bolsters, that the sections are rather thin, and I might remark, in the way of getting an elongation in our tests, which is considered a little low as compared with the other properties, that it is due to the fact that the chances of getting a sound test piece out of the casting are about one out of ten.

The reason why the tensile test results as shown in the table do not check well with those shown on the plate is that the figures in the table are from light castings, or bolsters, as stated, and the plate shows the figures as taken from heavy castings. We did get some good results from heavy castings. I might mention here one which I have noted. It happens to be a part of a locomotive frame. (By the way, we have some heat-treated locomotive frames in service, and they were quenched in

water.) It is about 0.30-carbon steel, double water quenched at 1,550° F., and the temperature drawn to 1,110°. This casting gave tensile results as follows: 51,000 lb. elastic limit, 90,000 lb. ultimate strength, a ratio of about 57, an elongation of 23.3 per cent., and a reduction of area of 41.1 per cent. We have duplicated such results as these on a number of occasions, and consider that they were very good.

In the matter of heating for 1 hr. for a foot thickness, I will say that this is considered by most people rather short, and I rather agree with Mr. Hall on that point. A good many of the mills take light castings, for instance, and let them soak for a period of about 4 hr. after the temperature is apparently up. For heavy castings some people allow 6 hr. The most vital thing in connection with heat-treating castings, especially those of irregular shape, weight, size, and form, is the time during which you have them in the water. This depends on the cross-sectional thickness and the irregularities of the shape. Four minutes are probably sufficient for water quenching of a casting of from 4 to 6 in., square or round, in cross-section. This gives sufficient chilling effect, and as long as the glow has absolutely disappeared, and the casting has steamed off and dried quickly when removed from the water, it is chilled and quenched. It should at that time be removed directly to the drawing furnace, before it is allowed to become cold. This eliminates the possibility of the introduction of any quenching stresses, and will also eliminate the chances of cracking.

Manganese Steel, with Especial Reference to the Relation of Physical Properties to Microstructure and Critical Ranges

BY W. S. POTTER, PITTSBURGH, PA.

(New York Meeting, February, 1914)

THE proportions of manganese and carbon in manganese steel are familiar to all, because manganese-steel castings have been well known for a decade or more in this country. The same alloy has now become a commercial product in the wrought state, and, as not only rails and bars are rolled daily but a great variety of forged and otherwise fabricated

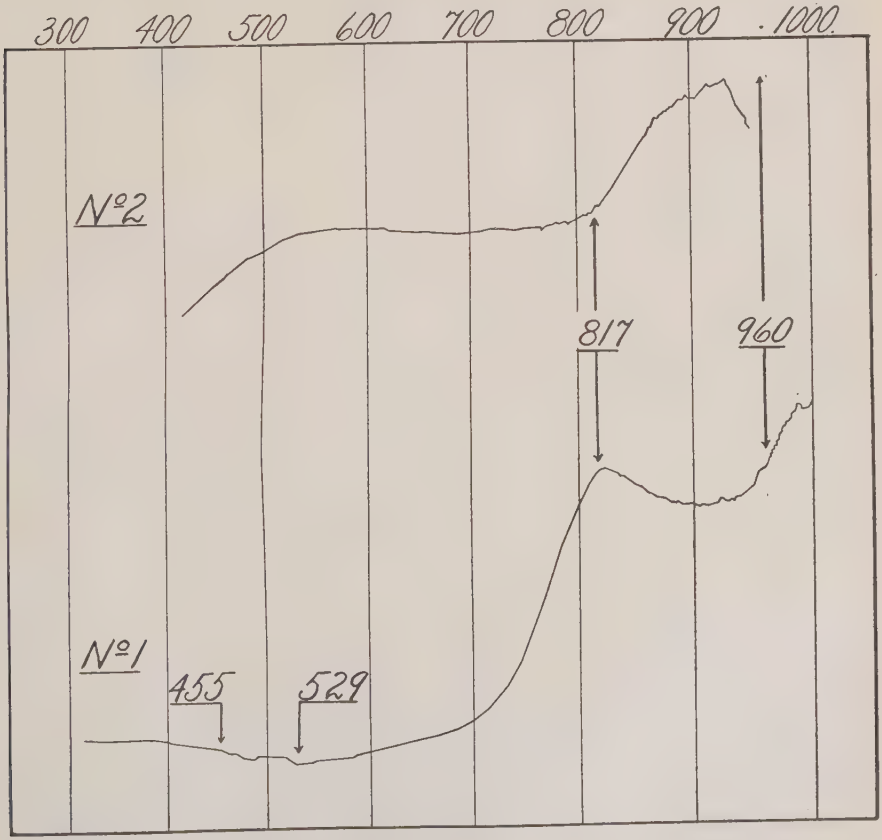


FIG. 1.—HEATING CURVES NOS. 1 AND 2. FORGINGS.

shapes are appearing, it may be interesting to mark the progress and, perhaps, to look a little closely into the nature of this important material of engineering.

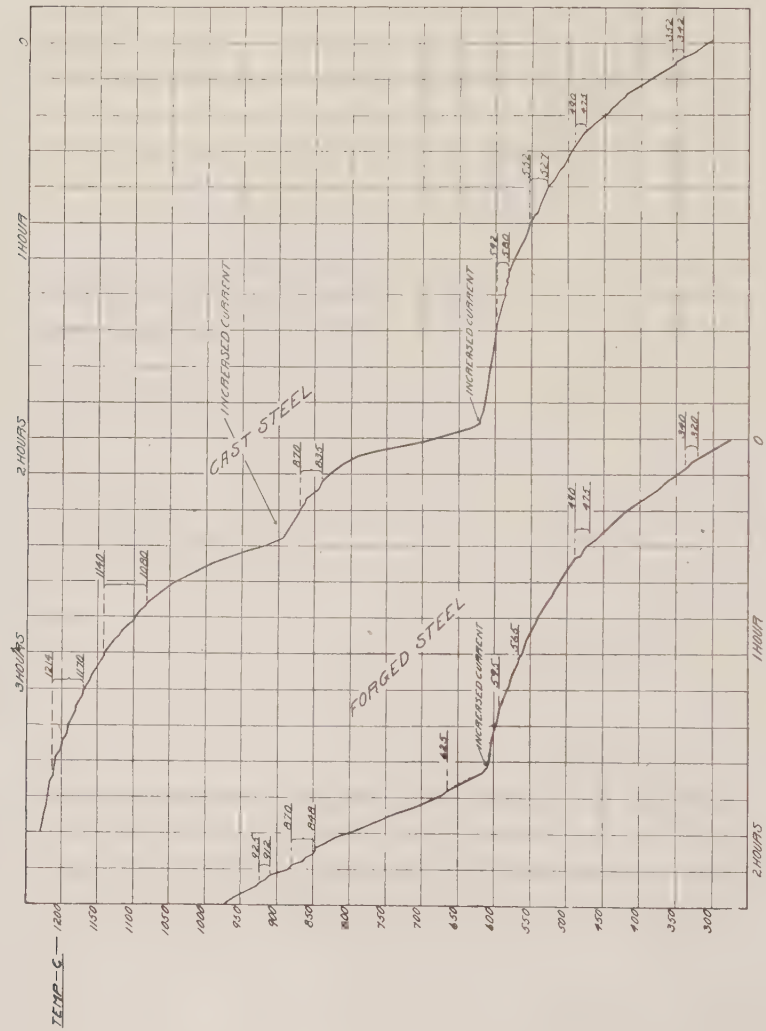
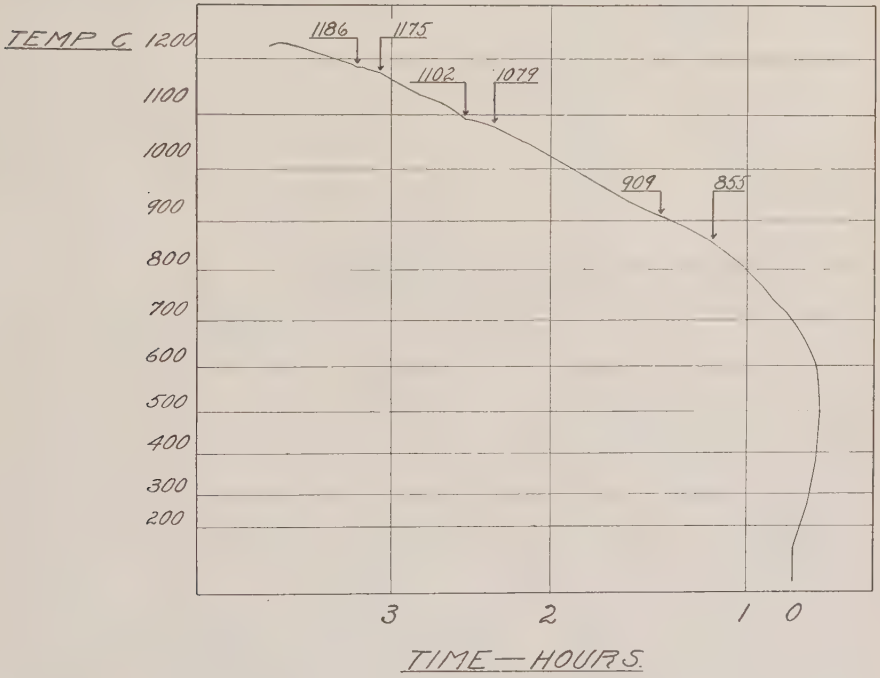


FIG. 2.—HEATING CURVE NO. 3. CASTING AND FORGING.

Accordingly, this paper sets forth briefly the results of a few tests and experiments and endeavors to connect the retardations, for the most part slight, observable in heating and cooling curves, with changes in microstructures seen after various heat treatments of the metal, and to correlate these with physical tests.

Retardations in Heating and Cooling

Concerning heating and cooling curves, it should be noted at the outset that when the steel has once been so treated that it has a uniform austenitic structure, it may be heated and cooled either slowly or rapidly between, say, 0° C. and its melting temperature (ordinarily at between 1,350° C. and 1,375° C.) and the retardations observable are slight, and without many repetitions, these might well be attributed to vibration or



Corrections: At 800° C. add 15°; at 900° C. add 17°; at 1,000° C. add 25°; at 1,100° C. add 30°; at 1,200° C. add 28°.

Fig. 3.—Heating Curve No. 4. Casting.
Crucible Melt No. 8. Mn, 14.15; C, 1.51; Si, 1.05

error of instruments. Specimens taken from small castings, and also rolled or forged pieces, generally show but slight retardations.

However, when the metal is first slowly frozen, and then slowly cooled —for example, at the rate of cooling prevailing in the interior of even a small mill size ingot—the retardations are much sharper and greater. As the manganese and carbon percentages are raised, the separable constituents are increased, also by slower freezing, and cooling, the separations are increased. With greater separations the retardations are naturally greater.

The structural constituents of the series of metals from 9 to 20 per cent. manganese, and from 1 to 2 per cent. carbon (probably also of alloys considerably outside this range) are gamma iron, or austenite; cementite, a double carbide of manganese and iron; a eutectic of cementite and austenite; and what appears to be a second eutectic of lamellar structure. Structures which have the appearance of martensite, troostite, and sor-

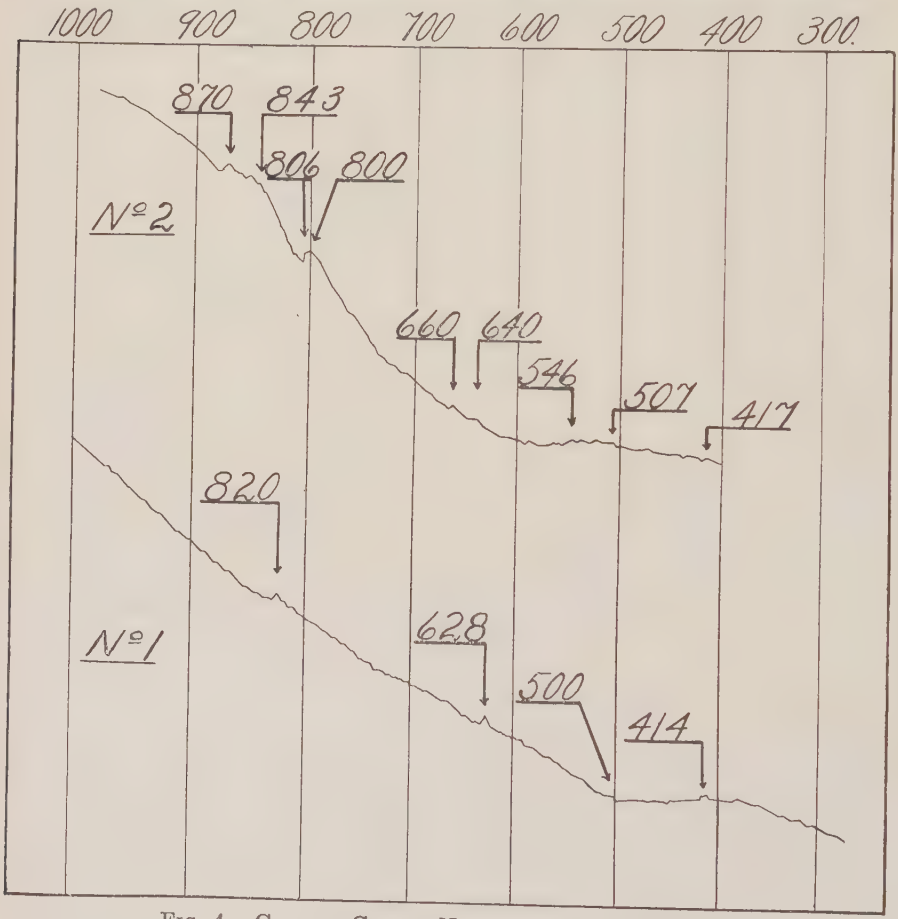


Fig. 4.—Cooling Curves Nos. 1 and 2. Forgings.

bite are also found in their proper relative positions. In cooling from the liquid, gamma freezes out as excess. Next, the lamellar eutectic freezes, although it is possible that the lamellar arrangement is due to mechanical break-up slightly below the end of freezing. Lastly, the residual liquid mixture of cementite and gamma iron freezes—this eutectic is white and is usually in part, or wholly, surrounded by the lamellar eutectic, although it sometimes appears within the austenite as a nodule from which the cementite radiates. As the temperature falls below 825° C. there is a

separation of the cementite from the austenite, giving rise to a variety of patterns, some decidedly martensitic. Troostite, and apparently sorbite, similarly result from the breaking down of the austenite as the temperature still further falls. On account of the very considerable difference in magnetic effects obtainable by cooling through, or heating in, different temperature ranges, it may be that the carbides separating in cooling at about 550° C., for instance, may not have the same composition as those

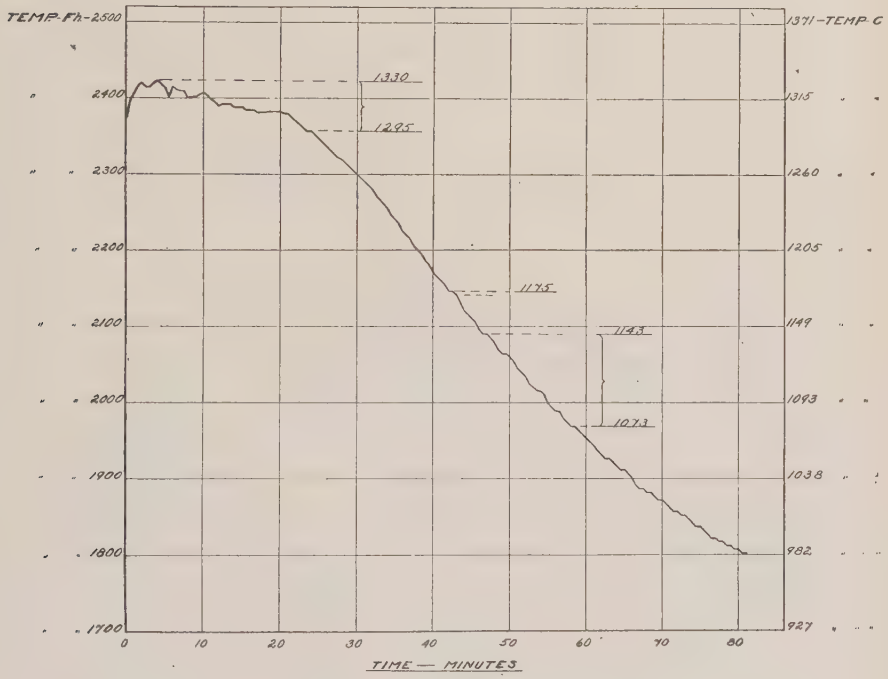


FIG. 5.—COOLING CURVE NO. 3.

Seventy Minutes from the End of Freezing at Point in Axis 30 in. from Top of 5,500-lb. Manganese Ingot. Mn, 9.60; C, 1.01; Si, 0.084. Usually the eutectic in this alloy occupies less than 1 per cent. of the area.

separating in martensitic form at 825° C., or those separating as cementite plates at about 1,250° C. In the reheated steel during slow cooling, between 1,300° and 1,200° C., cementite separates out of the austenite as plates and needles radiating from a central nodule.

The retardations in heating and cooling are shown in Cooling Curves 1 to 5, inclusive, and Heating Curves 1 to 4, inclusive, Figs. 1 to 7, and in the heating and cooling curves of Crucible Melts Nos. 6, 7, and 12, Figs. 8 to 10. Unless otherwise stated, the specimens examined have the following composition: Manganese, 11.00 to 11.50 per cent.; carbon, 1.10 to 1.15 per cent.; silicon, 0.05 to 0.10 per cent.

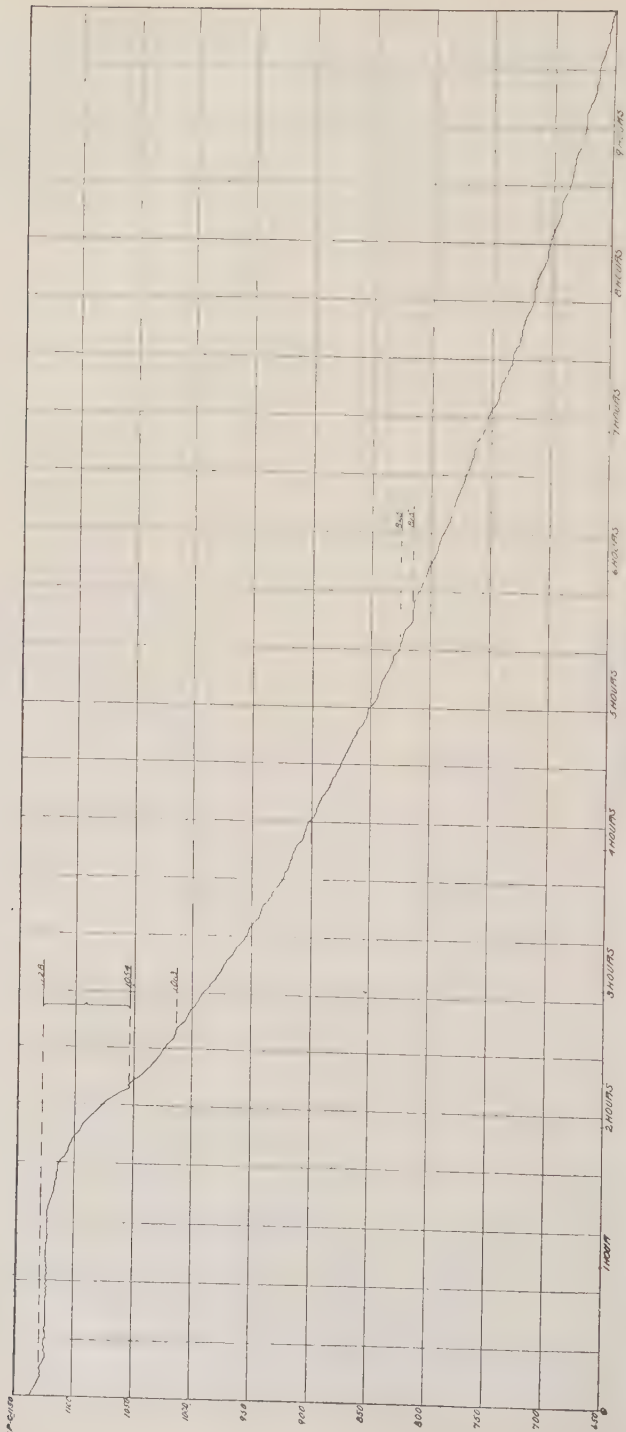


FIG. 6.—COOLING CURVE NO. 4, TAKEN FROM BEGINNING OF EUTECTIC FREEZING. CRUCIBLE MELT, 150 lb. Mn, 11.30; C, 4.17; Si, 0.56.

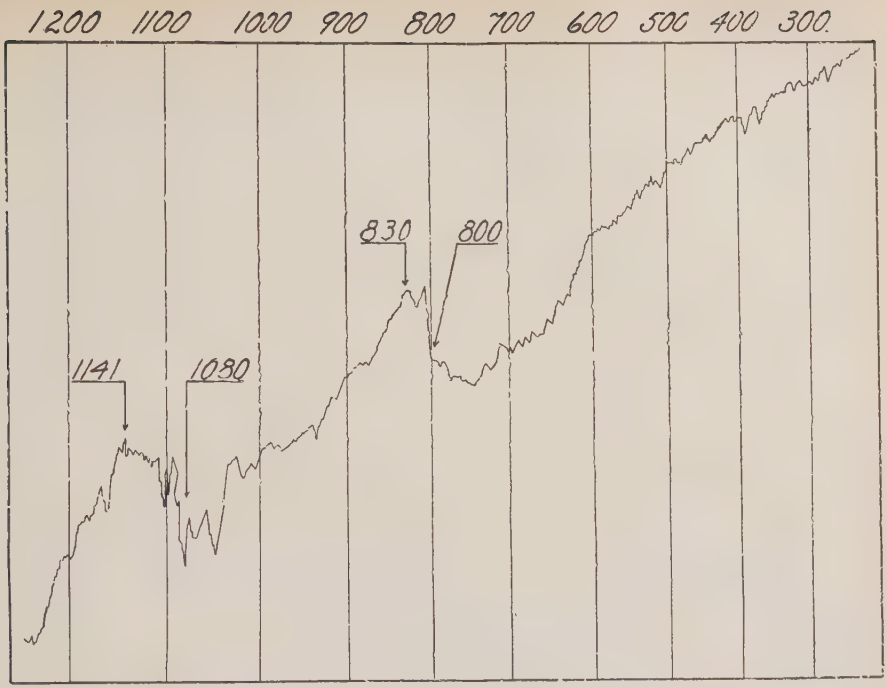
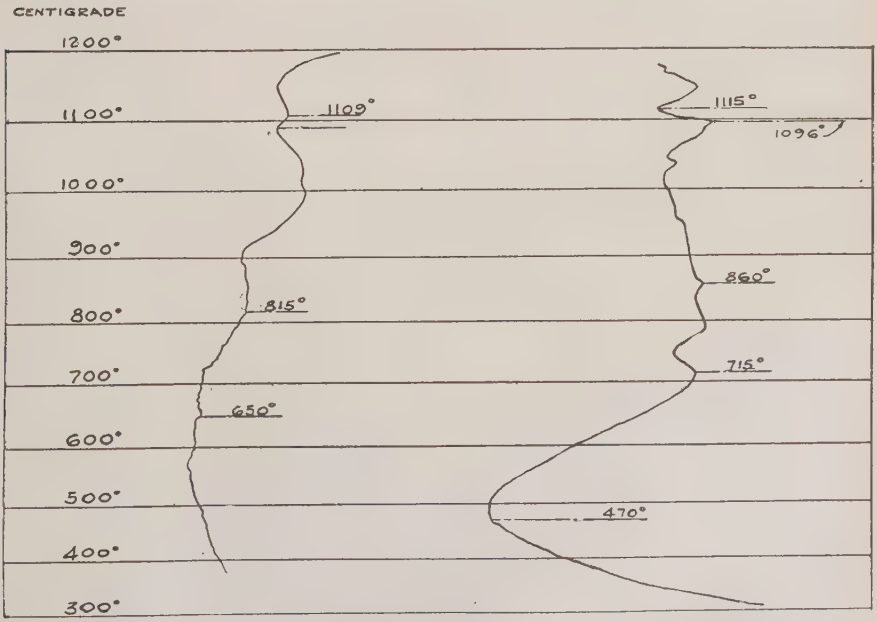


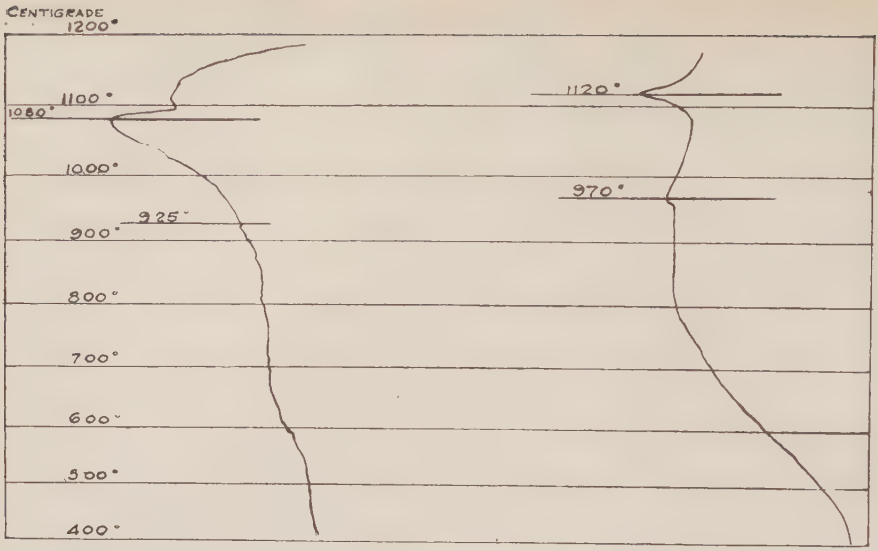
FIG. 7.—COOLING CURVE NO. 5. CASTING.

Specimen of Cast Ingot Metal. Mn, 11. 30; C, 1.11. Rate of cooling 300° C. per hour. This curve was made with Saladin apparatus on specimens taken from a rail mill size ingot, and shows the retardation due to freezing of eutectic, and the retardation corresponding to the breaking down of austenite.



First Cooling Curve. FIG. 8.—CRUCIBLE MELT NO. 6. First Heating Curve.

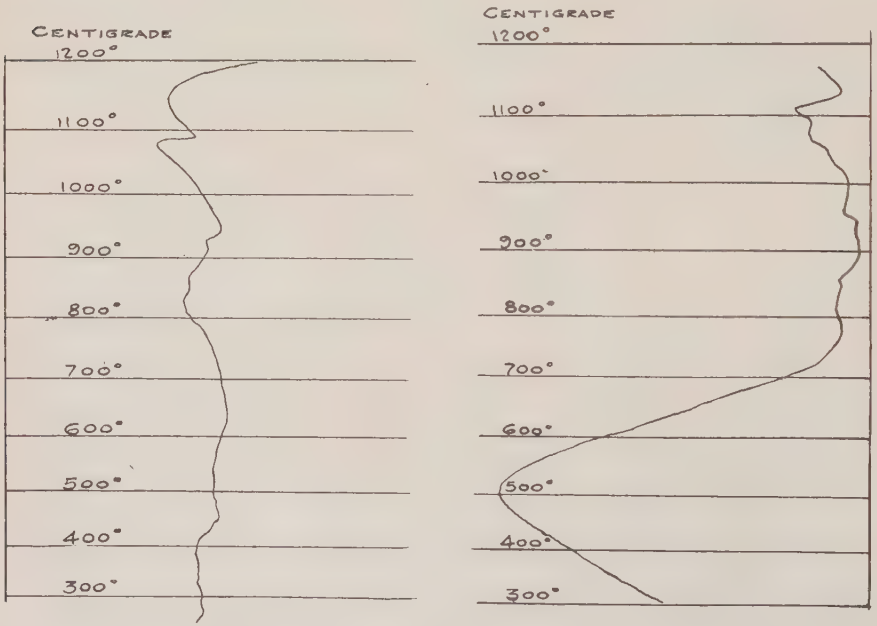
These curves were taken with Saladin apparatus and show retardation corresponding to melting and freezing of eutectic at about 1,100° C., re-formation and breaking down of austenite at about 850° to 825° C.: also the heating curve shows retardation corresponding to the break-up of austenite by reversion at about 500° C. Specimen was water quenched after cooling from the heat of casting to about 950° C. Beginning at about 1,150° C., retardation indicates the beginning of melting of the mass. Analysis o fspecimen was: **Manganese, 13.15; Carbon, 1.91; Silicon, 0.82 per cent.**



Cooling Curve. Heating Curve.

FIG. 9.—CRUCIBLE MELT NO. 7.

This specimen has analysis: Manganese, 12.54; Carbon, 1.56; Silicon, 0.46 per cent., and shows by Saladin apparatus the melting and freezing of eutectic.



No. 1 Cooling Curve. No. 1 Heating Curve.

FIG. 10.—CRUCIBLE MELT NO. 12.
Analysis: Mn, 25.35; C, 1.66; Si, 0.75.

In this very rich analysis there is more pronounced retardation between 900° and 1,000° C. than that seen between 800° and 900° C. Curves also show retardations corresponding to melting and freezing of eutectic, and in the heating curve there occurs the same retardation seen in heating curve of Crucible Melt No. 6, which is probably the break-up of austenite by reversion.

Conclusions from Heating and Cooling Curves.—The results of these heatings and coolings show:

1. That there is usually a well-defined retardation between 850° and 870° C., on heating, with corresponding retardation at 850° to 825° C., on cooling.

2. That a retardation at 1,100° to 1,140° C. on heating, is slight or well-defined, according to the composition of the alloy, and corresponds to a retardation on cooling at 1,125° to 1,060° C., which is equally dependent upon composition.

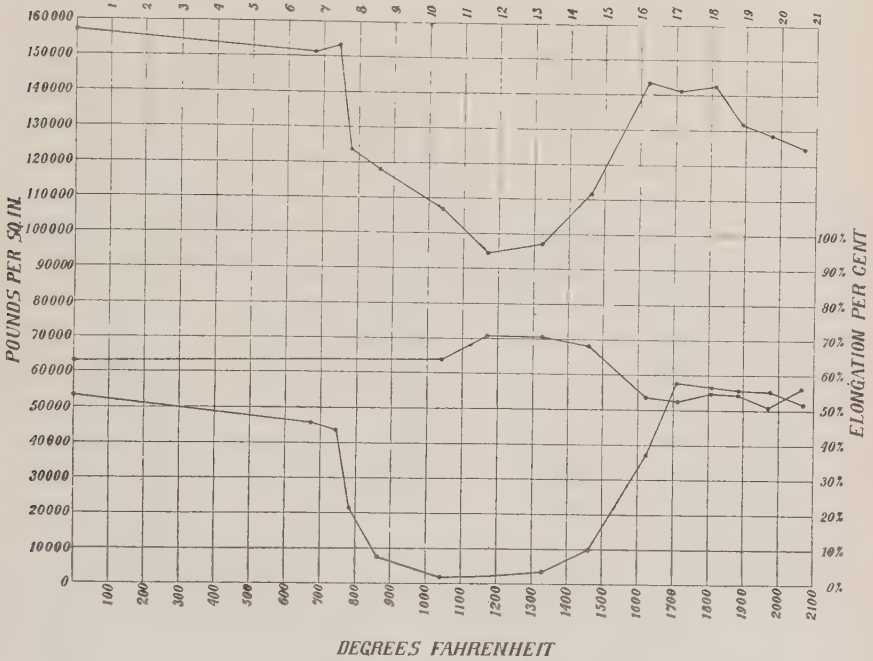


FIG. 11.—(SEE CORRESPONDING TABLE I).

3. That another retardation on heating at 1,200° to 1,210° C., which is also slight or well-defined according to the analysis, may correspond to slight retardations recorded during cooling, at 1,200° to 1,170° C.

4. That the retardation due to melting occurs at 1,350° to 1,375° C., and even lower with some alloys, while the retardations due to freezing occur at 1,370° to 1,320° C., and 1,300° to 1,275° C., respectively.

5. And, finally, that slight retardations on heating were recorded at the following temperatures, in degrees C.: 350 to 375; 450 to 470; 575; 675 to 735; 975 to 1,025; and, on cooling, at the following temperatures, in degrees C.: 1,010 to 1,030; 925 to 950; 650 to 710; 525 to 550; 420 to 450; 325 to 350.

Table I

(See corresponding Fig. 11)

Tensile Tests of Round Bars $\frac{3}{8}$-in. Diameter in Pulling Portion

Analysis of Steel: Manganese,13.00 per cent.; Carbon 1.02 per cent.
All bars quenched from heat of rolling.
Nos. 2 to 15 were also reheated for five minutes at the temperatures given, and then water quenched.

Treatment No.	Temperature, Degrees Centigrade	Elongation, Per Cent.	Breaking Strain, Lb. per Sq. In.	Elastic Limit	Remarks
1	.....	53.3	157,250	60,630	Crackle none or slight.
2	357	45.3	151,330	.....	Slight crackle.
3	394	43.7	153,720	.....	Slight crackle.
4	414	21.2	123,600	.....	No crackle.
5	460	7.8	118,000	.....	No crackle.
6	560	1.83	106,800	63,800	No crackle.
7	630	2.3	94,540	70,850	No crackle.
8	716	3.9	97,230	70,420	No crackle.
9	793	9.8	111,700	68,000	No crackle.
10	882	36.6	143,620	53,700	No crackle
11	932	57.5	141,040	52,640	Slight crackle.
12	985	56.4	142,450	54,500	Slight crackle.
13	1,029	55.6	131,690	54,430	Bad crackle.
14	1,076	55.7	128,590	50,690	Bad crackle.
15	1,130	34.4	124,780	55,700	Slight crackle.

The Relation of Heat Treatment to Tensile Properties.

The effect on tensile properties of treating manganese-steel alloys at different temperatures, and especially at temperatures where retardations were recorded in heating and cooling, is shown in Tables I to IV, inclusive, with the corresponding curves of Figs. 11 to 14.

Remarks on Table I and Fig. 11.—These test bars were reheated in lots of ten. The short time at the maximum temperature of reheating apparently did not give the steel quite time enough to come fully to equilibrium. In consequence, the relation of critical temperatures to effect of heat treatment is somewhat obscured. Also, the manganese percentage was higher than that of the bars of Series 2, 3, and 4, which would tend to raise slightly the temperatures corresponding, for example, to the breaking up of austenite, and to its re-formation. Therefore, the first great

drop in ultimate strength and elongation, instead of being found between 350° and 375° C., comes between 394° and 414° C. The next break corresponds to the retardation at 450° C. with a loss of 13.4 per cent. in the elongation and 5,000 lb. further loss of ultimate strength. The next decided drop in values occurs after passing from 560° to 630° C., with lowest elongation at 560° C., and lowest ultimate strength of the series at 630° C., thus giving good confirmation of the reality of the critical point which was observed as but a slight retardation in the heating curves at between

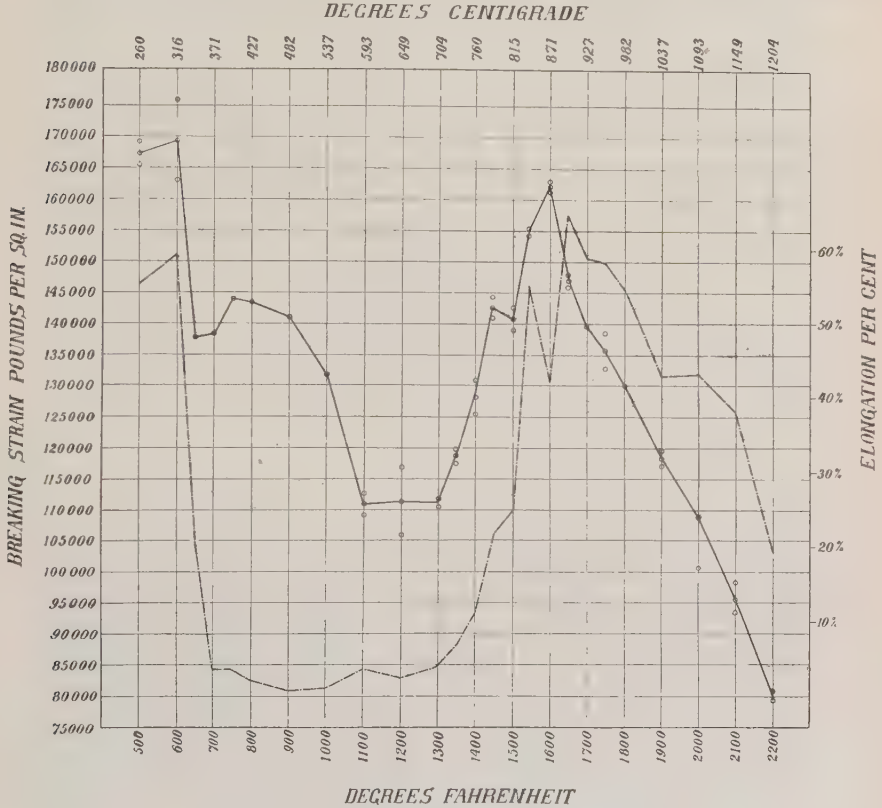


FIG. 12.—(SEE CORRESPONDING TABLE II).

Bars heated quickly to the temperatures given, held there for one hour, and then quenched in water.

550° and 535° C. The highest ultimate strength in the series for re-treated bars is at 882° C., the first heat treatment above the retardation noticed at 850° C. The percentage of elongation is greater in those specimens treated at temperatures higher than that which gave the greatest ultimate strength, but, in noting the persistence of high elongation up to lot No. 14, at 1,076° C., the delaying effect of short heating should be remembered, and the "crackle" beginning at 1,029° C. should be given due weight, as great elongation in a generally cracking bar does not indicate a good condition.

TABLE II

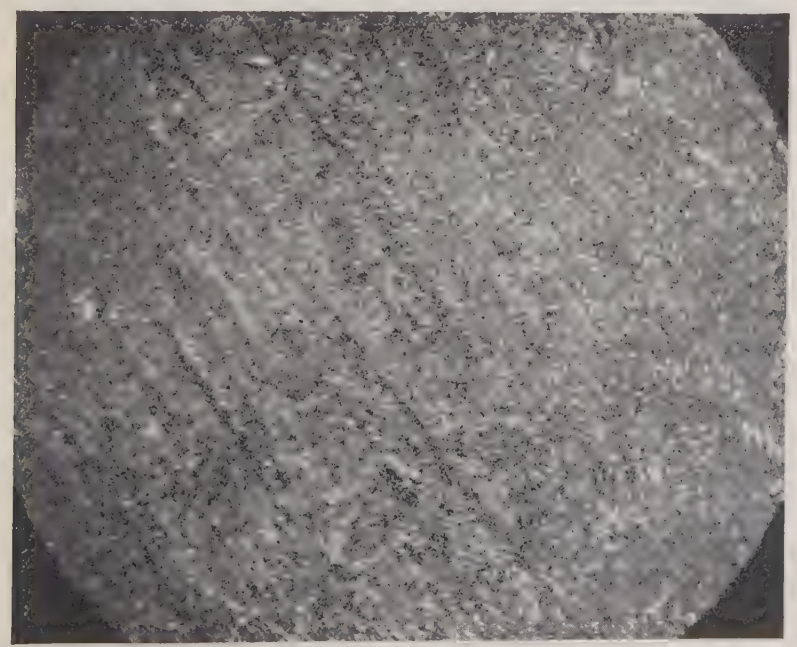
(See corresponding Fig. 12)

Tensile Tests on $\frac{3}{8}$-in. Diameter Round Bars. Heated to Temperatures Given and Water Quenched.

Treatment No.	Heated 60 Min. At	Elongation Per Cent.	Breaking Strain, Lb per Sq. Inch	Remarks	Micro No. All × 540
1	260	57.6	166,500	No crackle.	417
		53.0	169,300	No crackle.	
2	315	63.5	176,000	No crackle.	429
		55.0	163,100	No crackle.	
3	343	15.6	128,000	No crackle.	419
		21.7	137,700	No crackle.	
4	371	3.6	138,200	No crackle; uneven values.	420
		1.6	116,300	No crackle.	
5	399	3.7	144,100	No crackle; uneven values.	421
		1.0	112,400	No crackle.	
6	426	2.2	143,500	No crackle; uneven values.	422
		0.7	102,200	No crackle.	
7	482	0.8	141,100	No crackle; uneven values.	423
		0.6	129,100	No crackle; uneven values.	
8	538	1.7	121,000	No crackle; uneven values.	424
		1.2	130,200	No crackle.	
9	593	4.6	112,600	No crackle; nearly equal values.	425
		2.9	109,300	No crackle; nearly equal values.	
10	649	2.0	106,000	No crackle; nearly equal values.	427
		3.1	116,800	No crackle; nearly equal values.	
11	704	4.5	111,700	No crackle; nearly equal values.	428
		3.5	110,400	No crackle; nearly equal values.	
12	732	6.1	117,400	No crackle; nearly equal values.	432
		7.5	119,700	No crackle; nearly equal values.	
13	760	11.7	130,900	No crackle; nearly equal values.	461
		10.3	123,400	No crackle; nearly equal values.	
14	788	21.9	141,000	No crackle; nearly equal values.	442
		21.7	144,300	No crackle; nearly equal values.	
15	816	23.5	139,000	No crackle; nearly equal values.	443
		26.7	142,600	No crackle; nearly equal values.	
16	843	57.4	155,500	No crackle; nearly equal values.	444
		52.8	154,300	No crackle; nearly equal values.	
17	871	42.4	162,900	No crackle; nearly equal values.	445
		42.5	161,200	No crackle; nearly equal values.	
18	898	65.0	147,800	No crackle.	446
		63.8	145,900	No crackle.	
19	926	62.1	143,200	No crackle.	447
		58.9	139,400	No crackle.	
20	954	56.0	132,800	No crackle.	448
		60.3	138,400	No crackle.	
21	982	56.5	133,400	Slight crackle.	449
		53.2	132,100	Slight crackle.	
22	1,038	44.1	123,700	Bad crackle.	450
		42.0	121,200	Bad crackle.	
23	1,093	35.9	100,600	Bad crackle.	451
		50.8	117,300	Bad crackle.	
24	1,149	35.7	93,200	Slight crackle.	452
		40.9	99,200	Slight crackle	
25	1,204	17.3	80,600	Slight crackle.	453
		21.7	79,200	Slight crackle.	

Micrographs of Series 2 (*see Table II*).

Treatment No. 1, 260° C. Micro. No. 417.—Uniform fine austenite of the usual appearance of the steel when quenched from the heat of rolling, showing that, up to 260° C., the condition obtained by rolling and quenching is not altered. Acid attack slow and even.



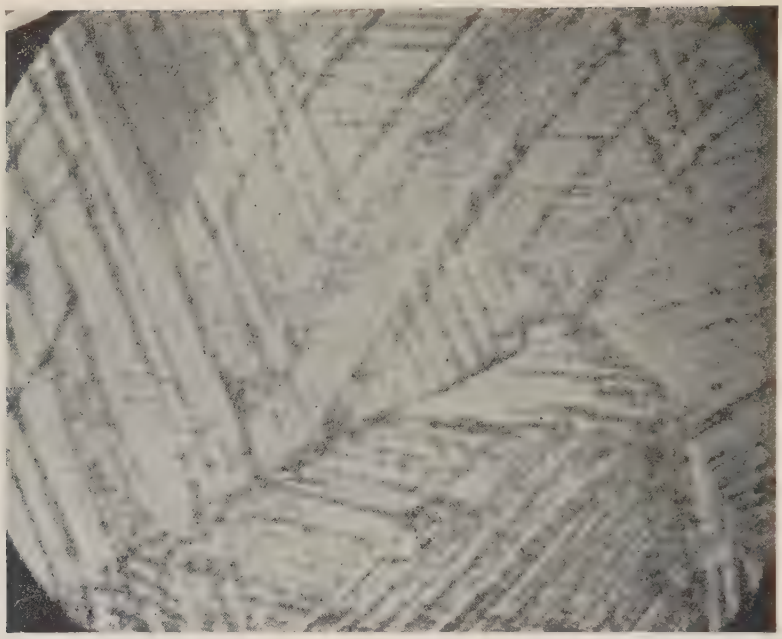
Treatment No. 5, 399° C. Micro. No. 421.—Austenite has disappeared and is replaced by a fine granular structure. A very fine network of lighter color beginning to form noticeably. Acid attack very rapid.

Treatment No. 7, 482° C. Micro. No. 423.—White network most pronounced. Acid attack very rapid.

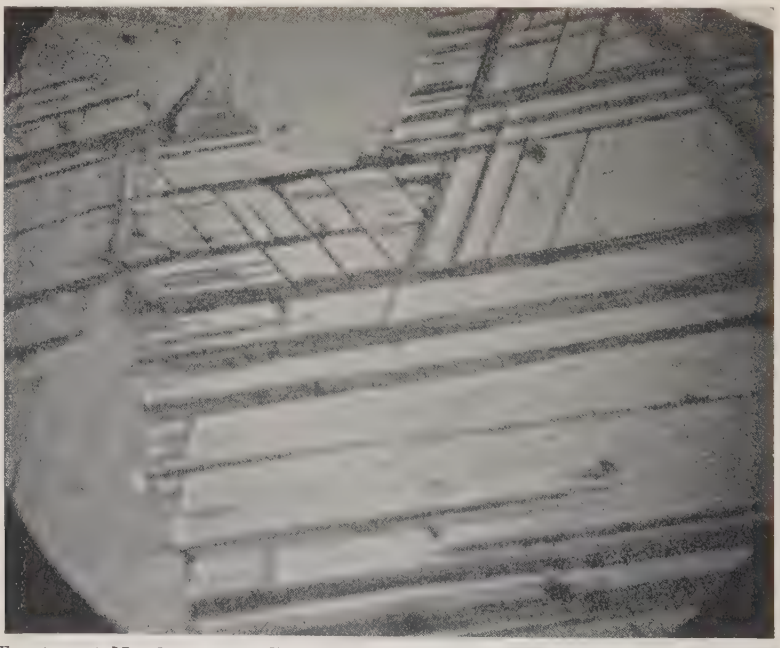
Treatment No. 9, 593° C. Micro. No. 425.—White network now very fine. Specimen has an appearance suggestive of moss. Acid attack rapid.

Treatment No. 16, 843° C. Micro. No. 444.—Austenite fairly well defined in places but not yet uniform. Acid attack very slow.

Treatment No. 19, 927° C. Micro. No. 447.—Large grains. Very characteristic coarse austenite. Good grain adhesion. Acid attack very slow.



Treatment No. 21, 982° C. Micro. No. 449.—Acid attack more rapid than preceding. Austenite breaking down at grain border, and etch deepest between grains.



Treatment No. 25, 1,204° C. Micro. No. 453.—Note remarkably close adhesion of grains, and grain outlines scarcely traceable, except by cutting off of austenite structure lines, and slip planes after 60 min. etching in concentrated solution of picric acid in alcohol.

Lots 11, 12, and 13, heated at 882° and up to 985° C., show much the best results.

Remarks on Table II and Fig. 12.—Note the sharp drop from 315° C. through 343° C. to 371° C. The lowest percentage of elongation appears after passing the retardation range at 450° to 475° C., with a high but irregular tensile strength from 343° to 482° C., while above 575° C., the ultimate strength, although lower, is much more uniform. Values rise after passing 704° C. and the best tests are from 843° to 982° C., the greatest strength being at 871° C., and the greatest elongation in the re-treated bar at 898° C. As the temperature rises above 898° C., the values fall in a fairly regular way, as might be expected, with rapidly increasing grain size and diminishing density.

In this series (Series No. 2), there are three temperature regions where the metal is uniform austenite:

First, when, after water quenching from heat of rolling, the bar is not reheated to a temperature much above 300° C.

Second, when the bar has been reheated to between 850° and about 950° C.

Third, when the bar has been reheated to a temperature between 1,125° and 1,225° C.

In the first two regions, physical tests show the steel to be at its best for strength and ductility.

Between regions one and two, the metal is weak and brittle.

Between regions two and three, the specimen, when bent or stretched, cracks generally between the grains, showing a very low grain adh esion, and in a manner suggesting that, at this time, some brittle substance must have separated from the grain and lodged at the grain surface. In fact, a close micro-examination often shows a fine white line between grains of steel heated between ranges two and three, while acid attack is always deeper at the grain outline than it is in either range two or three.

Temperatures included in region three cover substantially the range which has been found useful for the heating of ingots for rolling.

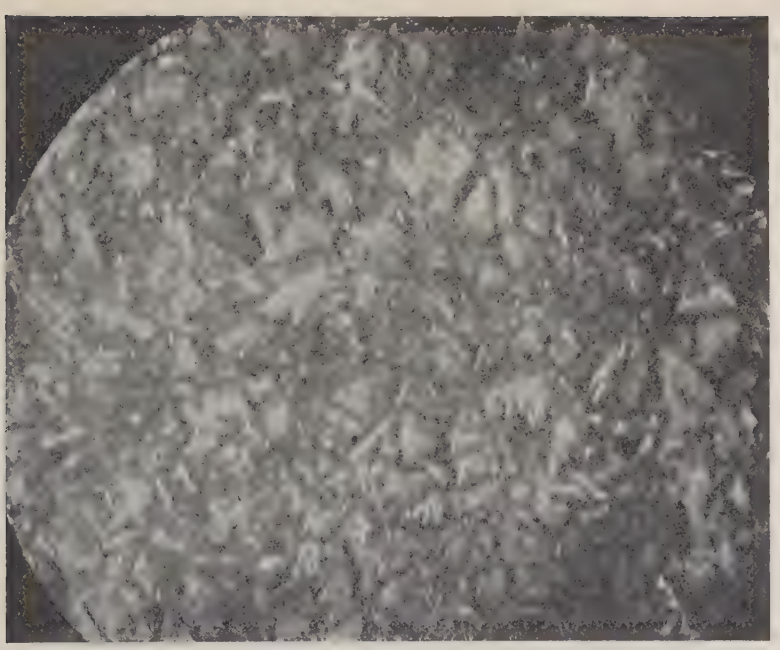
Temperature Range of Martensite Formation.

A diversion at this point may indicate the character of structure lying in the range between austenite ranges No. 1 and No. 2; that is, in the range of low-temperature heat treatments where austenite has been lost and not restored and where the steel is hardest and has but little ductility.

For this purpose six specimens of series No. 1 are compared with a specimen of the same steel heated to 1,325° C. and cooled slowly to ordinary temperature. The results are shown in the following micrographs:

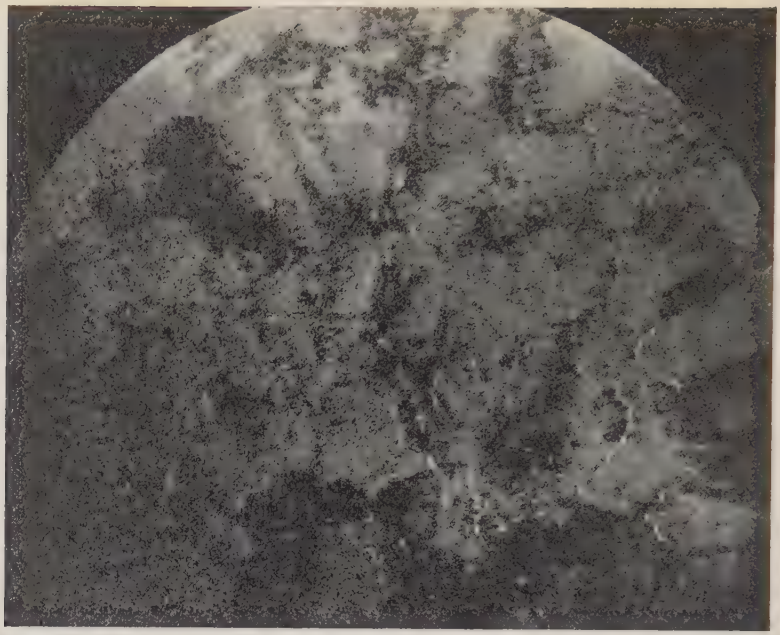
Micro. No. 13, 940 diam., quenched from heat of rolling bar. Uniform austenite. (Treatment No. 1, Series No. 1.)

Micro. No. 8, 470 diam., reheated 5 min. at 427° C. Unequal acid attack. Austenite disappearing. Network of carbides appearing. (Treatment between 4 and 5, Series No. 1.)

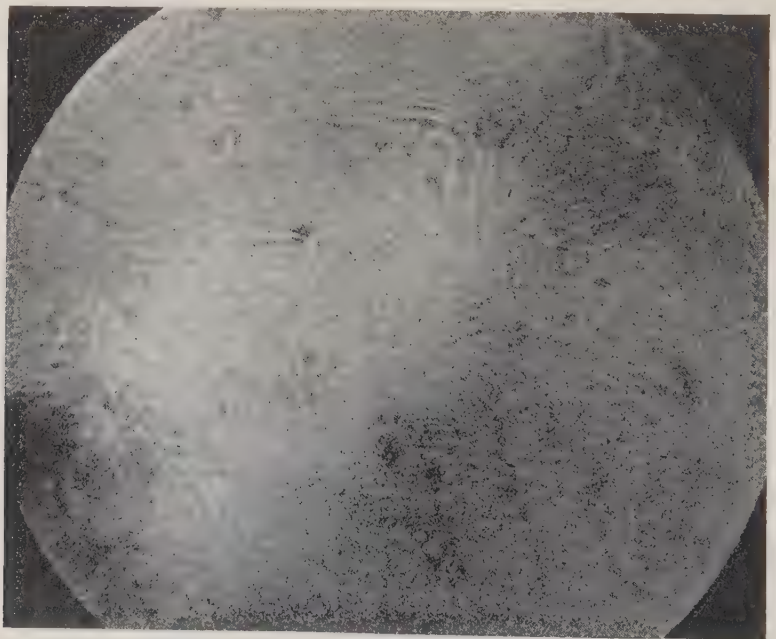


Micro. No. 34, 470 diam., 560° C., 5 min. etch. Austenite gone. Faint white network of carbides surrounding new grains having structure with fine white lines branching and intersecting at various angles. There is certainly room for the thought that this structure is martensite occurring in its proper position; that is, below the temperatures that produce austenite.

Micro. No. 36, 470 diam., 716° C., 15 min. etch. New structure forming within grains with large portion of surface still consisting of fine intersecting lines.



Micro. No. 37, 470 diam., 14 min. etch. Temperature 793° C. Resolution within grains almost complete; white network still showing at grain borders. High magnification shows new grain structure to be austenite.



Micro. No. 42, 470 diam., 31 min. etch. Temperature 895° C. Restored uniform austenite with no remnant of white constituent at grain outline.

Micro. No. 14, 950 diam. Slow cooled from 1,325° C. Carbide separation on grain border. Note form of white patches and grain structure resembling martensite.

Effect of Slow Cooling on Tensile Properties

The effect of slow cooling on manganese steel having 11 to 11.50 per cent. manganese and 1.10 to 1.15 per cent. carbon is shown in Tables III and IV and corresponding Figs. 13 and 14.

TABLE III

(See corresponding Fig. 13.)

Tensile Tests on ¾-in. Diameter Round Bars, All Heated to 1,149° C. and Cooled at Rate of 55° C. per Hour. Specimens Drawn and Water Quenched at Temperatures as Given

Treatment No.	Drawn from Furnace at	Elongation, Per Cent.	Breaking Strain, lb. per sq. in.	Remarks	Micro. No.
1	1,149°C.	25.6	83,380	No crackle.	.....
2	1,093	21.0	84,890	Bad crackle.	.....
3	1,038	29.4	85,260	Bad crackle.	.....
4	982	29.8	82,790	Slight crackle.	.....
5	926	27.1	81,210	Slight crackle.	.....
6	871	25.4	79,510	Slight crackle.	.....
7	816	26.7	80,720	Slight crackle.	.....
8	760	24.6	71,500	Slight crackle.	.....
9	704	26.5	82,090	Slight crackle.	.....
10	649	25.1	81,345	Slight crackle.	.....
11	593	1.0	34,380	No crackle.	431–540 diam.
12	537	0.2	33,620	No crackle.	434–150 diam. 430–540 diam.

Remarks on Table III and Fig. 13.—Bars 1 to 10, inclusive, broke in part through the grains, showing fairly good grain adhesion. Bars 11 and 12 showed no breakages of grains, the fractures being entirely between the grains; while micro-examination showed in bars 1 to 10 no separations between grains; and in No. 11 a slight separation of carbide, and in No. 12 a broad silvery band of separated carbides between grains with a breaking down of the metal at the edge of the grains.

Remarks on Table IV and Fig. 14.—Note that, during slow cooling from 926° to 843° C., there is no considerable change in physical properties, while a sharp break occurs between 843° and 816° C., corresponding to the location of the retardation point at about 825° C. in cooling. Also note that the lowest ultimate strength is found after passing from 538° to 510° C., corresponding to the slight retardation frequently noted at about 525° C. in cooling, and that the lowest percentage of elongation

appears after passing from 427° to 399° C., while a slight retardation is often found at about 425° C. Also, the ultimate strength is within 500 lb. of the minimum for the series after passing 425° C.

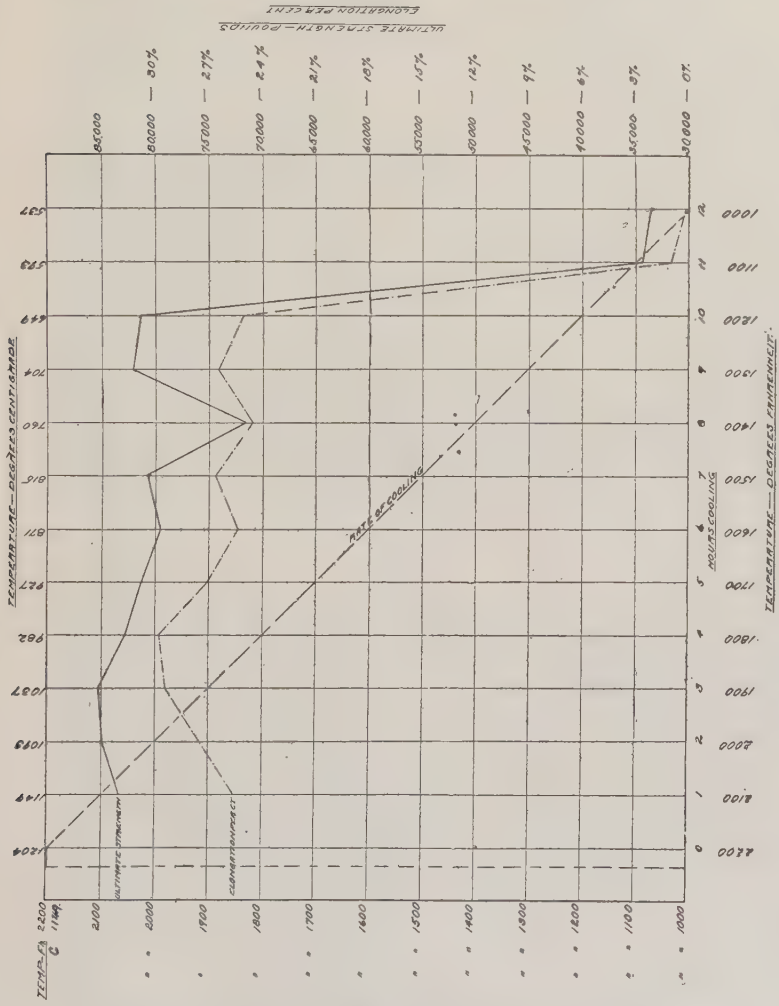
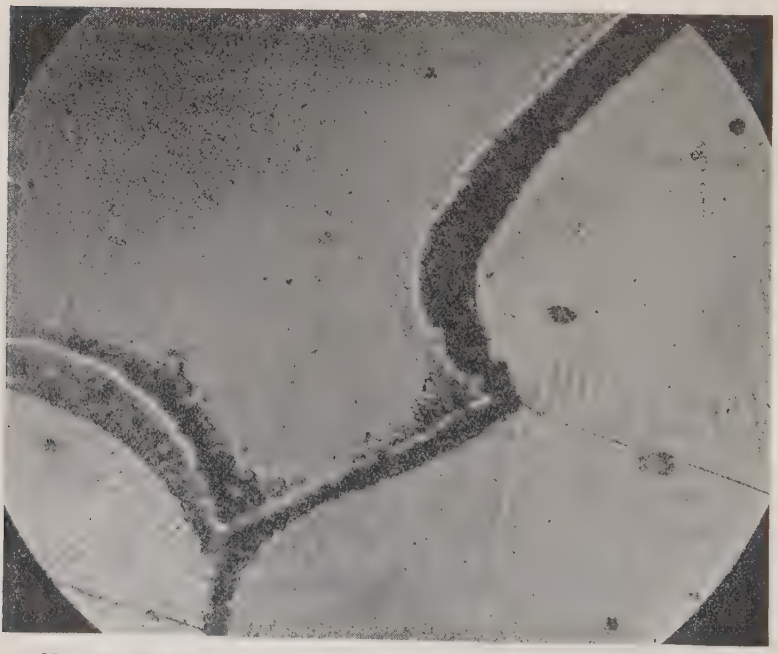


Fig. 13.—(See Corresponding Table III).
Twelve ¾-in. Round Bars heated quickly to 1149° C., held there for 20 min., then cooled at 55° C. per hour, and drawn and water quenched at temperatures noted.

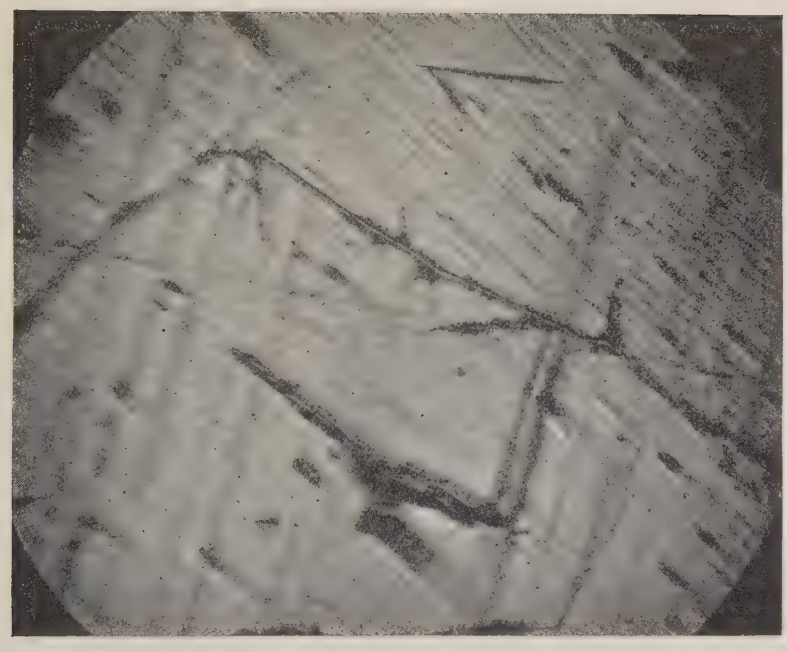
Conclusions as to the Metallography of Manganese Steel

I present the following as fairly well proved points for further observation: In steel containing 10 to 20 per cent. manganese, 1 to 2 per cent. carbon, during cooling, gamma iron begins to freeze at about 1,370° and is for the most part frozen at about 1,270° C.; the eutectic freezes at 1,125° to 1,080° C., or perhaps a little higher temperature with low carbon and manganese mixtures; the mechanical break-up of austenite begins at 850° to 825° C.; carbides separate between the grains at 710° to

Micro. No. 431, Bar No. 11, 540 diam. Slight separation.



Micro. No. 430, Bar No. 12, 540 diam. Large carbide separation. Note twinning.



Micro. No. 434, Bar No. 12, 150 diam. Large carbide separation.

650° C.; the magnetism increases at 550° to 525° C., with further carbide separation between the grains, and there is a further mechanical separation at 430° to 420° C.

In heating, if the steel is originally austenitic, there is a preliminary break-up at 350° C.; a further break-up at 450° to 470° C.; and a complete break-up at 575° C., with separation of magnetic carbides of iron or manganese at 475° and 575° C.

TABLE IV

(See corresponding Fig. 14)

Tensile Tests on $\frac{5}{8}$-in. Round Bars, all Heated to 926° C. Cooled at the Rate of 55° C. per Hour, and Drawn and Quenched at Temperatures Given

Treatment No.	Time in Furnace	Quenching Temperature	Elongation, Per Cent.	Breaking Strain, Lb. per Sq. In.	Remarks
1	1 hr.	926	51.6	153,800	No crackle.
2	1 hr. 30 min.	898	56.7	156,700	No crackle.
3	2 hr.	871	61.2	153,950	No crackle.
4	2 hr. 30 min.	843	64.2	153,200	No crackle.
5	3 hr.	816	44.8	137,200	No crackle.
6	3 hr. 30 min.	788	31.0	120,450	No crackle.
7	4 hr.	760	15.1	93,370	No crackle.
8	4 hr. 30 min.	732	15.7	88,750	No crackle.
9	5 hr.	704	10.5	84,120	No crackle.
10	5 hr. 30 min.	676	7.8	78,060	No crackle.
11	6 hr.	649	4.6	73,950	No crackle.
12	6 hr. 30 min.	621	4.5	76,940	No crackle.
13	7 hr.	593	4.5	78,040	No crackle.
14	7 hr. 30 min.	566	3.1	71,220	No crackle.
15	8 hr.	538	2.7	71,625	No crackle.
16	8 hr. 30 min.	510	1.5	68,070	No crackle.
17	9 hr.	482	1.4	74,440	No crackle.
18	9 hr. 30 min.	455	1.5	80,500	No crackle.
19	10 hr.	427	1.2	75,290	No crackle.
20	10 hr. 30 min.	399	0.5	69,570	No crackle.
21	11 hr.	371	0.8	72,500	No crackle.

In rolled steel: Separated carbides begin to be redissolved at about 725° C., and this is completed at 850° to 870° C., with re-formation of austenite.

Slight re-separation of carbides occurs along the grain outlines as the melting point of the eutectic is approached, beginning at about 1,025° C., and continuing to 1,100° C., or over. This is most noticeable in ingot

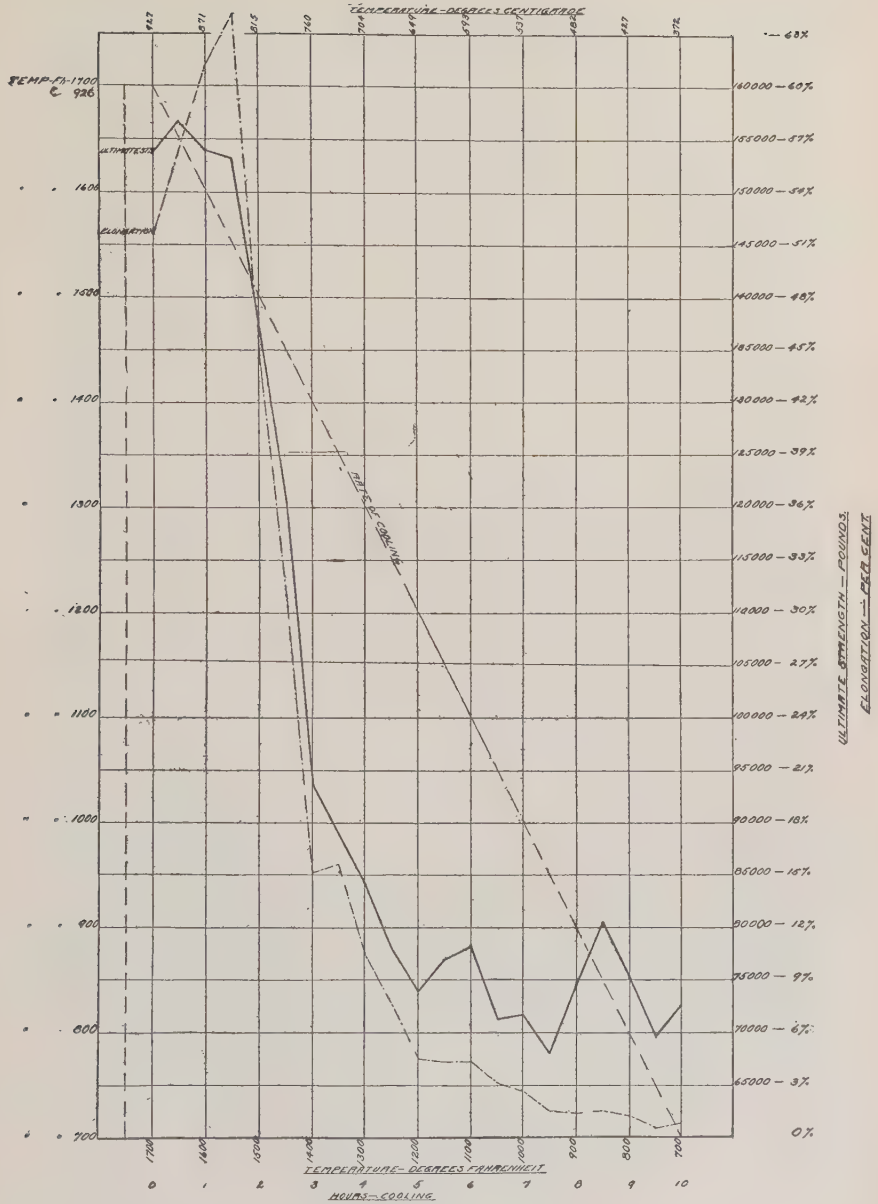


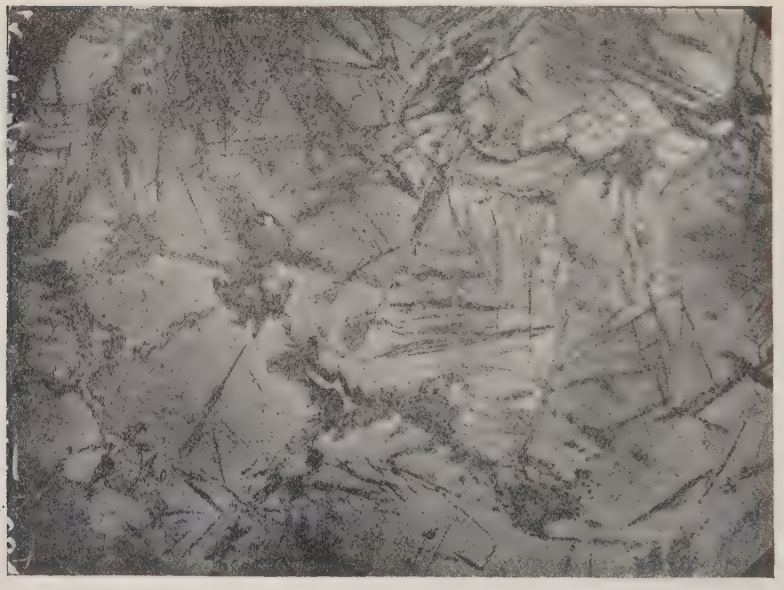
FIG. 14.—(SEE CORRESPONDING TABLE IV).

Twenty-one $\frac{5}{8}$-in. Round Bars heated quickly to 926° C., held there for 30 min., then cooled 100° F. per hour, and drawn and water quenched at temperatures noted.

metal, and taken together with the fact that the outlines of the original freezing structures persist in the cast steel at temperatures below 1,125° C. or thereabout, the remarkable tenderness of ingots between 1,025° and 1,125° C. is made clear.

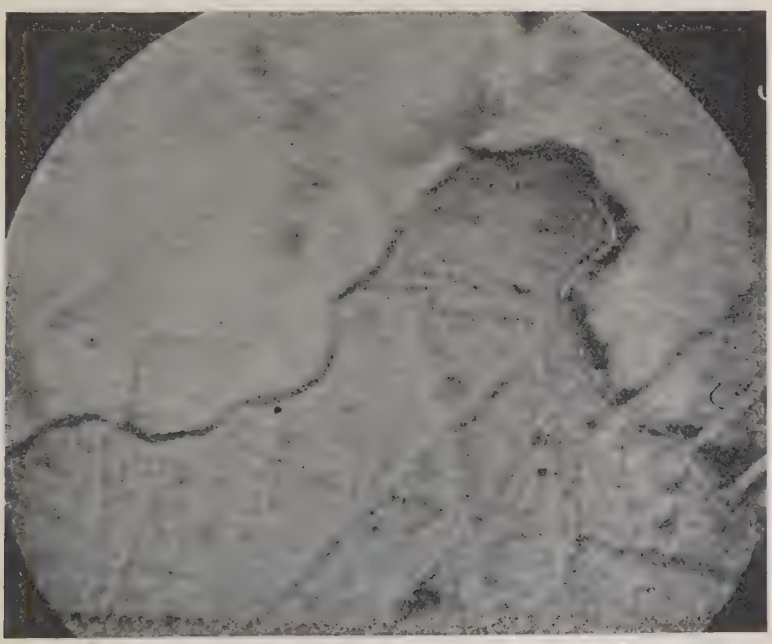
In cast ingot metal: The melting of the eutectic occurs at 1,120° to 1,140° C. if heating is slow, and at about 1,100° to 1,130° C. if heating up to this point is rapid; complete resolution of constituents and re-graining with first formation of uniform austenite takes place above 1,150° C.; cementite re-separates extensively above 1, 250° C.; the mass melts at 1,350° to 1,375° C., or lower, depending on its composition.

Structures in Cast Manganese Steel

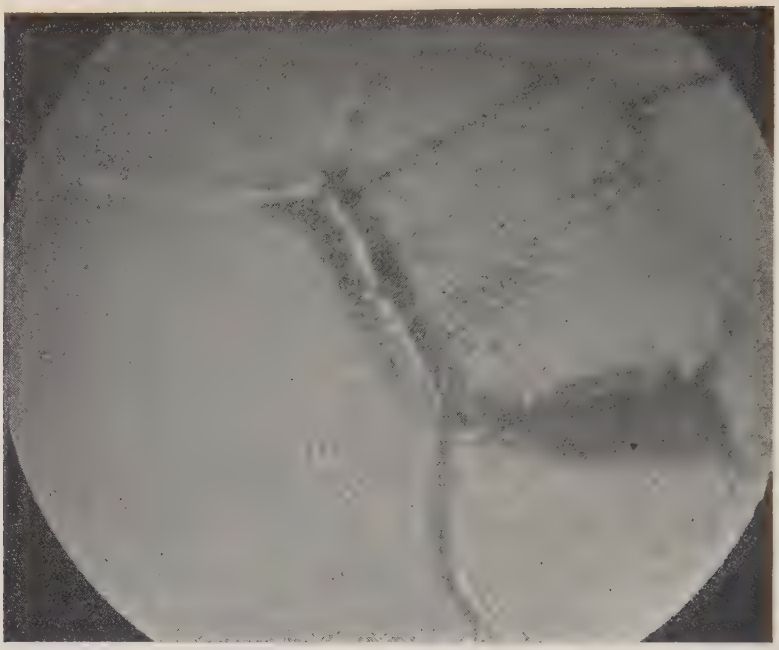


Micro. No. 466—90 diam. Shows a general view of structures in the interior portion of an ingot about 20 by 20 in. section. Cementite separations within gamma grains. Cementite and lamellar eutectics between gamma grains.

Micro. No. 457—540 diam. Same ingot. Another view of the inter-grain conditions. Eutectic and troostite with ground mass of gamma iron.

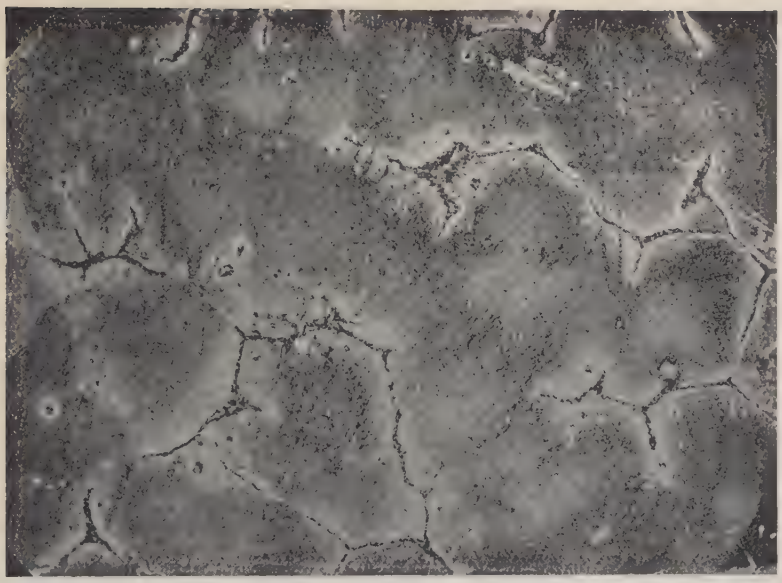


Specimen B. Micro. No. 488. Shows view 150 diameters of same ingot metal after slowly heating to 1,075° C., then holding for one hour at 1,075° C., then water quenching.



Micro. No. 487—540 diam. Same specimen as No. 488. Inter-grain conditions showing residuals of unabsorbed eutectic.

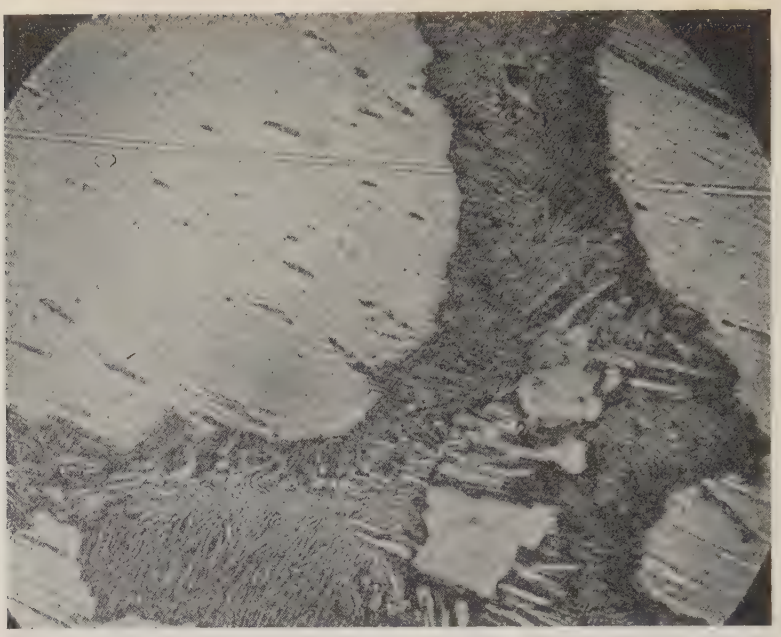
Specimen C. Micro. No. 515—540 diam. Same ingot as A and B, but reheated to 1,175° C., and held at 1,175° C. for one hour, then water quenched. Uniform austenite with close grain adhesion of re-formed grains.



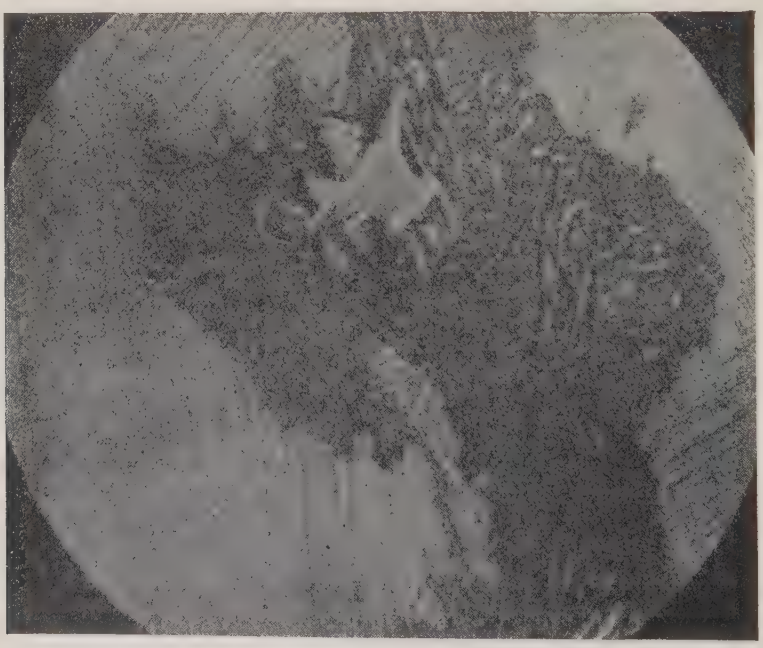
Specimen D. Micro. No. 462—90 diam. Specimen of same ingot as A, B, and C, heated rapidly to 1,250° C., and held for one hour at 1,250° C. Showing eutectic not absorbed.

Micro. No. 464—540 diam. Ground mass, gamma iron, with unabsorbed amorphous carbides and eutectic.

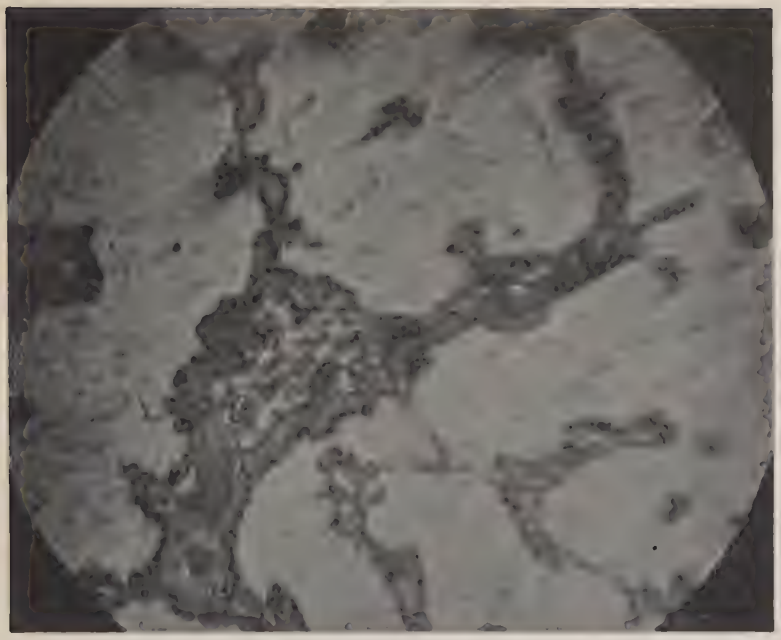
Absorption, that is, resolution, proceeds more slowly at 1,250° than at 1,175° C. for the reason that, at 1,250° C. and above, the cementite is separating.



No. 7 Crucible melt. Manganese, 12.54; Carbon, 1.56 per cent. Eutectic percentage higher than in Micro. No. 466. Micro. 485.—Cementite and eutectic of same appearance as noted in manganese-steel ingots of ordinary carbon proportion—a tough, ductile metal.



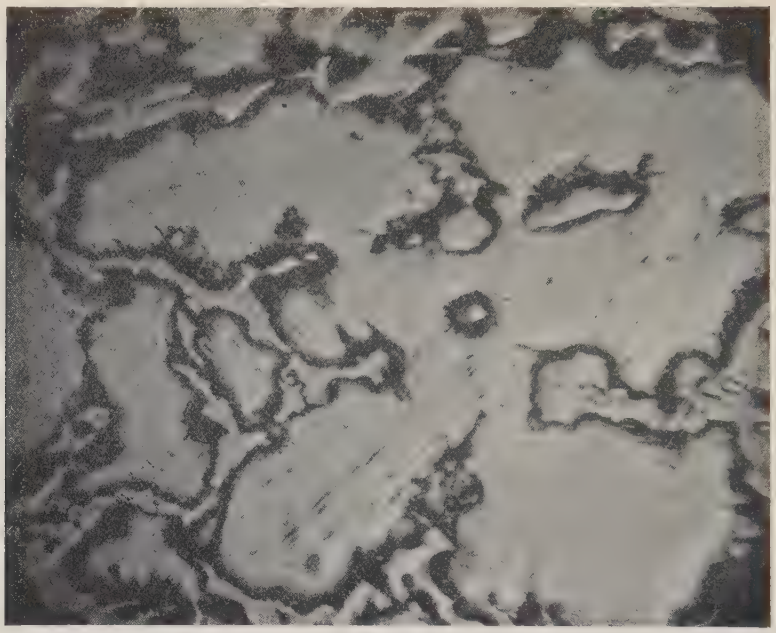
No. 8 Crucible melt. Manganese, 14.15; Carbon, 1.51; Silicon, 1.05 per cent. Micro. No. 566—540 diam. White and lamellar eutectic with ground mass of gamma iron and no martensite in gamma iron, because the specimen was quenched after cooling from freezing, to 1,050° C.—that is, just below the eutectic freezing range.



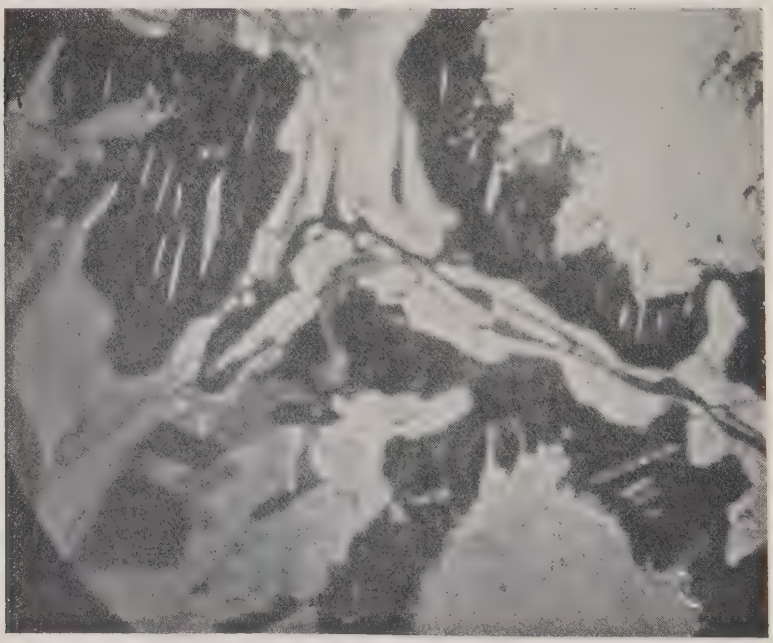
Micro. No. 565—80 diam. High eutectic percentage.

Crucible melt No. 9. Manganese, 30.58; Carbon, 2.05 per cent. Micro. No. 509 —540 diam. Coarse partly formed eutectic. Small portions of eutectic not frozen when specimen was water quenched.

This specimen was drawn from furnace after cooling from melting heat to about 1,100° C. and water quenched. At 1,100° C., metal was soft and broke in handling, showing that freezing was not complete, thus demonstrating that eutectic freezes within retardation range of 1,125° to 1,080° C.



Crucible melt No. 10. Manganese, 26.28; Carbon, 2.38 per cent.
Micro. No. 513—150 diam. Showing ground mass (unknown structure) with large cementite eutectic separation and only small area of a structure not typical lamellar eutectic, lying between ground mass and cementite. A weak, crumbling metal, decidedly not steel.



Micro. No. 514—540 diam. Showing ground mass doubtful, with large areas of cementite eutectic, and a eutectic differing from lamellar eutectic seen in typical manganese steel.

Discussion

Albert Sauveur, Cambridge, Mass.—While I do not think that Mr. Potter's conclusions in regard to the existence of several critical points in heating and cooling manganese steel are fully supported by the appearance of the curves exhibited, the results of his heat treatment and physical testing are most important and instructive. It seems to me that Mr. Potter has shown in a conclusive manner the breaking up of austenite on heating manganese steel, and that he has located accurately the limits of the range in which such breaking up takes place. I see in his results a confirmation of a view I have held for many years: namely, that the reheating of quenched manganese steel is, in reality, a tempering operation, even if it implies an increase rather than a decrease of hardness. Undoubtedly it represents a return to a more stable condition—the essential feature of the tempering operation.

Quenched manganese steel is austenitic; that is, its condition is that of a solid solution in which the iron is in the gamma form. This is proven by (1) its structure, (2) the absence of magnetism, (3) the absence of critical point during the cooling of such steel, (4) its moderate hardness, high tensile strength, high ductility, and relatively low elastic limit. On reheating the metal to about 500° C., the austenite begins to break up through the formation of some beta iron, as evidenced by increased hardness and greatly decreased ductility. At some 500° C. some alpha iron also forms, since the metal becomes magnetic. The behavior of manganese steel when subjected to heat treatment appears to me to fall in line, in a remarkable manner, with the allotropic theory of the hardening of steel as first outlined by Osmond. The fact that, on tempering austenitic manganese steel, its hardness *increases* affords, in my opinion, a very strong evidence of the existence of a hard variety of iron between the soft alpha variety and the moderately hard gamma variety; in other words, of beta iron.

Sir Robert Hadfield, London, England (communication to the Secretary.*)—I have read with much interest Mr. Potter's paper on Manganese Steel, and compliment him upon his excellent photomicrographs.

In view of his opening remarks, in which, no doubt unintentionally, he states that manganese steel has only been known and introduced for about a decade, it seems to me he hardly does justice to the many willing helpers who have worked long before the decade of which he speaks, on manganese steel, which I discovered now about 30 years ago. When, therefore, he speaks as if this material, with its many wonderful qualities, has only been in use for the short space of 10 years, he evidently does so

* Received Mar. 27, 1914.

without being aware of the real facts. Manganese steel, both in its cast and forged forms, has been commercially used for more than 20 years. Therefore the statement referred to hardly seems to do justice to those in England and America who have been working so patiently and for so long a period and trying to master the intricacies of this interesting metallurgical product.

As a matter of interest to those of our younger members who are coming to the front in metallurgy, it may be interesting to make a brief mention of those to whom credit has been due.

The first in this country to take an interest in manganese steel was the late Joseph D. Weeks, of Pittsburgh, a very able writer and well acquainted with much of the practical side of metallurgy. He read the first paper in this, or in fact in any country, on my material, before this Institute in 1884. He met my father and myself in Europe just at the time the development of this steel was being made, and was so much interested that he asked if he might say something about it in America; hence the paper in question.

Not long after, Prof. H. M. Howe came over to Europe, and was so much struck that he also asked if he could speak of the research work we were then carrying on. As is well known, he gave a considerable and lengthy account of manganese steel in his great work, *The Metallurgy of Steel.*

Then came my friends the late W. J. Taylor and Maunsel White, also H. D. Hibbard, each of whom did much valuable work on manganese steel now some 23 years ago. Professor Howe read several papers regarding my discoveries, including those before the American Society of Mechanical Engineers in 1891, and the Franklin Institute in 1892. As I have just said, he also devoted considerable space to describing and commenting upon the material in his well-known *Metallurgy of Steel*, published 25 years ago. Since then, younger members of the Taylor Co., including Knox Taylor and others, have also worked indefatigably at the problem of developing and improving manganese steel during the last 15 years.

Among others in this country who did pioneer work with regard to the material are C. W. Hubbard, Pittsburgh; the late respected William Metcalf, of the Crescent Steel Works, Pittsburgh; also his firm, Miller, Metcalf & Parkin, all of whom took a practical interest in the subject; P. H. Dudley; Mr. Harrington, mechanical engineer of the Pittsburgh, Cincinnati & St. Louis Railway; S. T Wellman, of the Otis Steel & Iron Co., who made some of the earliest mechanical tests; E. G. Spilsbury, one of the first to experiment with manganese-steel wire; T. T. Morrell, chief chemist of the Cambria Iron Works; J. P. Shinn, also his brother; the Lehigh & Wilkes-Barre Coal Co.; and many other individuals and corporations.

It may be of interest to also add that among those who took an interest in developing manganese steel on the continent of Europe were: in France, M. Saladin, of Messrs. Schneider; Chatillon & Commentry Co.; Messrs. Gautier, Osmond, Pourcel, Dumas, H. and A. le Chatelier, and many others. In Germany, Messrs. Krupp and their Directors, Dr. Schmidt and Dr. Ehrensberger; Professor Ledebur, Dr. Muller, and others.

I have only mentioned a few of those who have been interested in doing research and practical work on manganese steel. There are, of course, many others, and I have expressly avoided referring to those concerned in this matter in Great Britain, in this reference to what has now become a world industry.

As a landmark and date for reference, it may be mentioned that the first manganese steel I ever produced, as duly registered in my experimental records, was in March, 1882.

The work done and the interest taken by those mentioned—all men of well-known metallurgical reputation—took place not a decade ago, but in most cases from 15 to 20, or even 25 years ago. It is only fair that their work should not be forgotten. While we are glad to have new investigators in the field of research, those who have done good work in the past—alas, a number of them not now with us—should not be overlooked.

With reference to the remarks about grain adhesion,—we would prefer to use the term "grain cohesion"—our experience, going back over experimental and practical commercial work for 25 years, has been somewhat in the opposite direction to that of the author, and we believe that the well-defined etching crystalline structure is evidence of good-quality steel.

As regards "crackling," this we have often noticed in test bars, and we consider it to be due to the cracking of the decarbonized surface skin. It will be noticed this chiefly occurs in bars heated to the higher temperatures, which supports our theory.

The author refers to three temperature regions where the metal is uniformly austenitic. We consider that above a certain temperature, probably about 1,000° C., uniform austenitic structure exists. In other words, above 1,000° C. the cementite is all dissolved, resulting in a single uniform constituent of austenitic structure.

As regards the remarks on p. 453, the author states that in region three the temperatures of 1,125° and 1,225° C. have been found useful for the heating of ingots for rolling. A research was made regarding the temperatures used for rolling ordinary carbon-steel rails, and it was found that the regular practice ran in some cases to as high as 1,170° C. or more.

Some of the photographs shown by the author appear to be covered

with parallel lines. It is probable that these parallel markings are Neumann lines (slip bands), and are due to the fact that manganese-steel sections are somewhat difficult to prepare. In fact, owing to their toughness, if they are knocked or hammered off a larger piece, this disturbs

FIG. 15.—NEARLY PURE AUSTENITE OF QUENCHED MANGANESE STEEL. × 50.

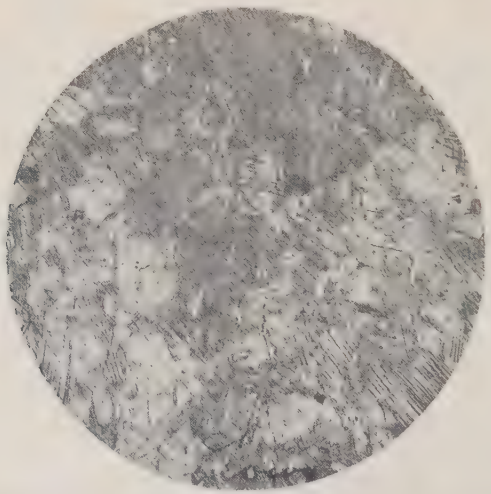


FIG. 16.—AUSTENITE CHAINED BY HAMMER BLOWS. SHOWS NEUMANN LINES IN THE CRYSTALS (Slip bands). × 150.

the cleavage planes of the austenitic crystals. We have produced such a result, as shown by the accompanying photomicrographs. Fig. 15 represents pure austenitic structure, and Fig. 16 is taken from a part of the same section after having been knocked and hammered off.

The latter shows the parallel lines due to the mechanical disturbance of the austenitic cleavage planes caused by hammering. They appear more numerous in this photograph than those shown by the author because the magnification is less, but on the other hand a larger field is shown, and it is interesting to note that some of the crystals have stood the treatment quite unmarked.

As regards the effect of heat treatment, it may be of interest to give the result obtained from one of my earliest specimens of manganese steel.

These tests were made in 1887, and are of interest as showing the effect of heat treatment, namely, (A) as forged; (B) afterward heated to a high yellow and cooled in air; (C) cooled in oil; (D) cooled in water. It will be noticed the elongation varied exactly in accordance with the relative conductivity of the medium in which the specimen was cooled.

It is also of interest to note that this was the first bar of steel ever produced in the world's history which gave the exceedingly high tenacity of 67.13 tons per square inch accompanied with 44.44 per cent. elongation, *and yet the specimen still remained unbroken.* These tests were carried out by the British Admiralty Inspector, in the year 1887, details being given in the following table:

Tests on Hadfield's Manganese Steel

Composition: C, 1.26; Si, 0.28; S, 0.05; P, 0.11; Mn, 14.01 per cent.

Mark	Treatment	Breaking Load Tons per Square Inch	Elongation on 8 in., Per Cent.	Appearance of Fracture
A	Forged into test bar and no further treatment	36.43	1.56	Fine grain
B	Heated to high yellow and cooled in air	47.90	14.06	Granular; fine grain
C	Heated to high yellow and cooled in oil	55.04	26.56	Fibrous
D	Heated to high yellow and cooled in water	67.13	44.44	Fibrous[a]

[a] The Admiralty Inspector who carried out tests on this bar remarked: "Notwithstanding 44.44 per cent. elongation, and a tensile load of 67 tons, this bar is still *unbroken.*" It is probable that the final elongation would have been 51 to 52 per cent., with a breaking strength of a little over 70 tons per square inch.

The author of the paper states that the maximum results are obtained at 840° C., but we have often obtained both superior tenacity and equal if not superior elongation at temperatures in some cases as high as 940° C., or over 100° C. higher than those mentioned by the author.

Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel

BY PROF. B. HOPKINSON, F. R. S., AND SIR ROBERT HADFIELD*, F. R. S., LONDON, ENGLAND

(New York Meeting, February, 1914)

Introduction.—A short time ago Professor Stoughton asked the writer if he would present a paper for the February meeting of our Institute. In reply to this suggestion, some notes have been prepared relating to the subject mentioned in the title. It is to be hoped they may be of interest to our members.

The research now described is the joint work of Prof. B. Hopkinson, F. R. S., and the writer. Those interested in the subject will find further data in a paper entitled The Magnetic and Mechanical Properties of Manganese Steel, which is to be read at this year's May Meeting of the Iron and Steel Institute in London. This paper was submitted to the Institute in October last, and advance copies will shortly be available.

The present paper has been drawn up by the writer on this side the water, and he has not had time to communicate with Professor Hopkinson with regard to the statements or opinions expressed herein. While the writer believes that Professor Hopkinson would in the main agree with the statements herein set forth, it should be understood that apart from the actual work described he is not otherwise responsible.

The experiments described have been carried out on the material known as manganese steel, which one of the writers of these notes discovered in 1883. A paper by him describing this material was read before this Institute in 1893, entitled Iron Alloys, with Special Reference to Manganese Steel. It is not necessary to describe manganese steel nowadays; its qualities are so well known, and its use throughout the world is on a very extensive scale.

The Early Use of Manganese Steel.—The writer always looks back and with pleasure thinks of the many willing helpers in this country, who by their encouragement and advice greatly assisted him in testing and introducing manganese steel. As was inevitable with a new material, many practical difficulties had at first to be overcome. Without the help now referred to, and which was so willingly rendered, it would not have been possible to make the great strides which have been accomplished in such a comparatively short space of time. Manganese steel, outside carbon

*Honorary Member.

steel, which has been used for centuries, was practically the first alloy of iron with another element to be produced on a large scale. It will be admitted that since then the work done in this field of alloys of iron with other elements has been of incalculable benefit to the world generally.

The Properties of Manganese Steel.—Probably no other material presents so many curious properties for examination and research as manganese steel. It is hard, and yet it is extremely tough when properly treated; it is naturally non-magnetic, and yet it can be made magnetic; its electrical resistance is very high; its permeability is exceedingly low; in the ordinary way it cannot be machined, yet by proper heat treatment, this difficulty can to a large extent be overcome; in fact many of the specimens referred to in this paper are quite readily machined. Yet the same product in its non-magnetic condition can only be machined with difficulty. Its hardness is most peculiar, because while having only about 200 Brinell ball hardness in its toughened condition, yet it is then almost as difficult to machine as chilled iron. It is very fluid, more like cast iron, yet sets rapidly. Its melting point is comparatively low as compared with mild or medium carbon steel. Although so hard when cold, it can be readily forged, rolled, pressed, or otherwise manipulated while hot.

The Hardness of Manganese Steel.—As regards the hardness of manganese steel, the following experiments may be referred to.

In 1904, one of the writer's licensees rolled a manganese-steel rail. This rail was experimented upon as follows. After 20 min. drilling, 11 g. of manganese steel were removed, while in the same space of time 750 g. were removed from an ordinary carbon-steel rail. A piece of the same manganese-steel rail bent double cold. Its tenacity was 60 tons per square inch, with an elongation of 40 per cent. This is shown in Fig. 1 of the paper on Manganese-Steel Rails, presented at this meeting, and is interesting not only because of the date at which the rail was produced, nearly ten years ago, but because it brings out the point to which the writer wishes to call attention: namely, that notwithstanding the apparently extreme hardness of manganese steel to drilling and other similar tests, its ball hardness is only about 200. In other words, its hardness may be described as "tough hardness," in contradistinction to the brittle hardness of hardened carbon steel with ball hardness numbers of 600 to 800.

Manganese-steel axles and tires were forged and rolled under the writer's direction in 1893. Notwithstanding their extreme toughness under the drop tup tests, they possessed the same peculiar "toughness hardness" to the drill, though the ball hardness was still about the same as that mentioned above for the rail: namely, from about 190 to about 210. The mechanical tests carried out on the foregoing are fully referred to in the writer's paper to this Institute in 1893, previously mentioned.

Making Manganese Steel Magnetic.—In the same paper and the same

date—that is, 1893—the writer pointed out the fact that manganese steel submitted to long heat treatment in an annealing furnace, on cooling down was found to be magnetic. It should be added that it has now been found that this appearance of magnetic quality in non-magnetic manganese steel was not due to the fact that it had been heated to the temperature prevailing in the annealing furnace, namely about 800° C., but that upon cooling down the specimen had passed through the critical range of about 500° to 550° C., which produced magnetic qualities. This treatment is referred to more fully on p. 488 of this paper.

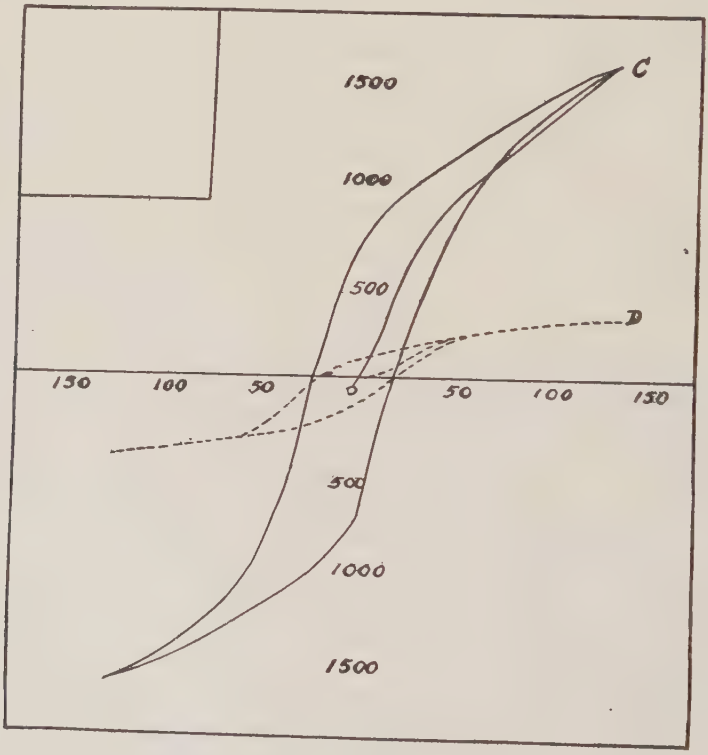


FIG. 1.—HYSTERESIS CURVES.

This was further shown in the writer's paper on The Results of Heat Treatment on Manganese Steel and Their Bearing upon Carbon Steel, read before the Iron and Steel Institute in 1894. A specimen of manganese steel, marked "C," heated and cooled in a cementation furnace, became considerably magnetic. Dr. Hicks, F.R.S., of the University, Sheffield, was good enough to make for the writer hysteresis curves of this cemented specimen, shown by the full lines in Fig. 1.

A specimen of the same manganese steel, marked "D," heated and cooled in a cementation furnace, then water quenched, while magnetic,

was much less so than specimen "C." This specimen possessed the hysteresis qualities shown by the dotted curve in Fig. 1.

Another specimen of the same material, marked "F," submitted to cementation, was then forged and water quenched. This became practically non-magnetic like the original specimen "A." In a field of 325 C.G.S., the permeability was 1.08, that is about the same as manganese steel in its non-magnetic condition and not submitted to cementation.

The above information is fully set forth in the following Table I.

TABLE I.—*The Effect of Cementation upon Manganese Steel*

Mark on Bar	Treatment	Magnetic Qualities	Remarks
A	As rolled.....	Not attracted by the most powerful magnet.	In a field of 320 C.G.C. the permeability = 1.061
B	Similar to "A," but water quenched at about 1,100° C.	Same as "A."	
C	Cemented. No subsequent heat treatment.	Considerably magnetic.	See full curve "C" in Fig. 1.
D	Same as "C," but water quenched.	Much less magnetic than "C."	See dotted curve "D" in Fig. 1.
F	Cemented, forged, and then water quenched.	Practically non-magnetic and like specimen "A."	In a field of 325 C.G.S. the permeability = 1.08.

NOTE: Sample "A" was a specimen of ordinary non-magnetic manganese steel (1.20 per cent. C., 13.42 per cent. Mn.), and from it were prepared, with the subsequent treatment named, specimens B, C, D, and F.

The electrical tests were carried out by Dr. Hicks, F. R. S., of the University, Sheffield.

As stated at the commencement of the writer's remarks, this paper is chiefly intended to present the facts with regard to the relation of the specific magnetism and the hardness of manganese steel on specimens submitted to different kinds of heat treatment. As already mentioned, the more comprehensive paper on this subject, which has been before the Iron and Steel Institute since October last, contains full description of the methods of determining the specific magnetism, taking the heating and cooling curves, photomicrographs of the specimens experimented upon, and other details, so that it is unnecessary to describe them here.

Effect of Treatment on the Non-Magnetic and Magnetic Qualities of Manganese Steel.—As is well known, among some of the many curious features met with in alloys of iron with other elements, are their behavior with regard to magnetic qualities. Manganese steel, while non-magnetic, can be made magnetic. Some of the iron and nickel alloys are

also non-magnetic, but by certain treatment—for example, immersion in liquid air—can be made magnetic. While the present joint research by Professor Hopkinson and the writer relates to experiments which have been in progress for about three years with regard to the effect of heat treatment on manganese steel, it may be mentioned that the writer in 1894 read a paper before the Iron and Steel Institute, referred to on a previous page, in which it was pointed out that by certain heat treatment manganese steel could be made magnetic, also that this quality could be reversed at will; in fact, by means of some of the treatments now described, which produce magnetic qualities, the material can be made so that it can be tooled. A bar of manganese steel which is non-magnetic, either in its brittle or tough condition, can by heat treatment be made magnetic; in fact, its specific magnetism can be increased until it is about half that of Swedish charcoal iron, and no doubt by still longer heat treatment in the specimens now referred to, the specific magnetism can be increased until it is about 60 to 70 per cent. that of Swedish charcoal iron. It may also be of interest to add that the writer has been able to produce in the same bar, one portion absolutely non-magnetic with a mere trace of specific magnetism, and the other end, by heat treatment, so magnetic that the magnet would readily lift it. In the latter condition, the specific magnetism is about 42 per cent. as compared with Swedish charcoal iron.

The experiments referred to in the writer's paper in 1894 aroused a good deal of interest at the time, many of the leading metallurgists taking part in the discussion which arose with regard to the facts noticed in the research in question.

Experiments on Alloys of Iron and Nickel.—Experiments have also been carried out upon alloys of iron and nickel, and these are also shown to possess reversible qualities, but by quite other treatment than that which effects the changes in manganese steel from non-magnetic to magnetic conditions.

The Effect of Low Temperatures.—Increase of temperature renders manganese steel magnetic, but does not apparently affect non-magnetic iron and nickel alloys, or at any rate to the same extent. Low temperature, that is, liquid air temperature, renders the iron and nickel alloys magnetic, but has not the slightest effect upon manganese steel even if the treatment is continued for as long as 18 hours. These are curious facts.

Some confusion has arisen because the non-magnetic alloys of iron and nickel, containing about 22 per cent. of nickel, become magnetic by immersion in liquid air. By some unaccountable means, some scientists came therefore to the conclusion that non-magnetic manganese steel possessed similar qualities. This, however, is incorrect. Manganese steel entirely differs from the iron and nickel alloys in the fact that it

cannot be rendered magnetic by immersion in a medium having even lower temperature than that of liquid air. In other words, it does not seem possible to render manganese steel magnetic by reducing temperatures to very low points in the thermometric scale. The only manner in which it can be made magnetic appears to be the heat treatment shown in the writer's paper of 1894, previously referred to, and also by the treatments now described, which have been made during the last three or four years.

The Results of the Research.—As the subject appears to be one of interest, Tables II and III, showing the various analyses, tests, and other information, are submitted for consideration.

Various specimens of manganese steel of standard type, and from the same billet, were submitted to heat treatment at the temperatures set forth in Table II, which gives the duration of heat treatment in hours, the nature of the medium in which the cooling occurred, the specific magnetism, and the Brinell hardness number of each specimen before and after treatment. In order to afford some idea of the mechanical qualities of the material, there are also shown the results of the bending tests upon specimens $\frac{5}{8}$ by $\frac{3}{8}$ by $1\frac{1}{2}$ in. between the supports. The maximum load given is the breaking load, and the angle of bend is given in degrees. The mechanical tests also show in a remarkably clear manner the difference in physical characteristics between the various specimens, varying from bent double cold, to bends so small as to be scarcely measurable.

Table II shows the magnetic qualities and certain mechanical tests, including ball hardness numbers, of manganese steel in its non-magnetic and magnetic conditions. The results have been arranged in three groups:

(A) Completely or practically non-magnetic (less than 1 per cent. magnetism).

(B) Partly magnetic (1 to 12 per cent. magnetism).

(C) Very magnetic (20 per cent. magnetism and over).

In Table III the same figures have been arranged according to temperatures.

It will be seen that while the specific magnetism of the specimens shown in groups (A) and (B), with the exception of those in group (B) which were heated to 600° C. and the one specimen heated for 48 hr. at 450° C., is very little, certain differences present themselves. The specimens in group (A) may be termed practically non-magnetic; in group (B) there is a tendency for the specific magnetism to increase; in group (C), representing treatment between the critical temperatures of from about 500° to 600° C., the specific magnetism is quite considerable, varying from 20 to 40 per cent. as compared with 100 for Swedish charcoal iron having a purity of 99.82 per cent. Fe. As before mentioned, with

TABLE II.—*Magnetic and Mechanical Tests upon Manganese Steel in its Non-Magnetic and Magnetic Conditions*

Note 1: Analysis of the material:

C	Si	S	P	Mn
1.21	0.22	0.030	0.080	12.36

Note 2: "S.C.I." refers to Swedish charcoal iron of about 99.82 per cent. purity Fe.

Test Piece No.	Specific Magnetism, S.C.I. = 100	Brinell Hardness Number	Bending Test ¾ by ¾ by 1½ in. between Supports		Temperature in Degrees Centigrade	Specimen Cooled Down in	Duration of Heat Treatment, Hours
			Corrected Load, Maximum in Lb.	Angle of Bend, Degrees			
			"A" Completely or Practically Non-magnetic (Less than 1 per cent. Magnetism)				
	0.3	207	.....	180 unbroken	.....	Quenched from yellow heat in water.	No further treatment
5857	0.3	215	10,750	161	300 310	Furnace....	168
5970	0.3	267	10,900	21	750	Air.....	1
5971	0.3	277	12,180	29	750	Water.....	1
5973	0.3	277	12,890	29	530 750	Furnace... Water....	15 1
5298	0.2*	337	10,360	8	450	Furnace....	6
5972	mere trace	339	10,920	11	530 750	Furnace.... Air.....	15 1
5402	0.2	340	12,900	12	400	Furnace....	6
5296	0.2*	408	10,080	2	650	Furnace....	3
5303	0.2*	418	9,747	2	650	Furnace....	6
5403	0.6	340	11,680	8	400	Furnace....	12
			"B" Partly Magnetic (1 to 12 per cent. Magnetism)				
5872	1.25	217	10,630	124	300	Furnace....	552
5856	1.0	317	14,400	26	300 400	Furnace.... Furnace....	384
5855	1.0	328	12,900	17	300 400	Furnace.... Furnace....	384
5400	1.0	364	11,400	6	400	Furnace....	24
5389	1.2	361	10,770	4	450	Furnace....	12
5399	2.0	387	9,700	2	400	Furnace....	48
5373	2.2	387	9,250	3	450	Furnace....	24
5374	7.0	418	8,270	2	450	Furnace....	48
5749	12.2*	444	7,420	1	600	Furnace....	3
			"C" Very Magnetic (20 per cent. Magnetism and over)				
5661	19.8*	418	8,476	1	600	Furnace....	12
5748	20.0	418	8,478	1	550	Furnace....	6
5750	20.4*	351	12,570	11	500	Furnace....	6
5336	24.4*	375	11,130	5	600	Furnace....	24
5658	24.8*	402	11,801	4	550	Furnace....	12
5304	25.8	438	10,140	2	550	Furnace....	24
5369	28.8	351	9.970	3	500	Furnace....	12
5713	29.8*	430	10,270	2	600	Furnace....	72
5327	30.0*	418	11,270	3	600	Furnace....	48
5330	31.4*	495	8,037	1	550	Furnace....	48
5338	34.4	444	7,890	1	500	Furnace....	24
5337	40.2*	438	7,460	1	500	Furnace....	48
5364	42.0*	444	8,200	nil	500	Furnace....	600

* Repeat tests were carried out on these specimens, similar results being obtained.

TABLE III.—*Magnetic and Mechanical Tests upon Manganese Steel in its Non-Magnetic and Magnetic Conditions*

Arranged According to Temperature

Temperature in Degrees Centigrade	Duration of Heat Treatment, Hours	Specific Magnetism S.C.I. = 100	Brinell Hardness Number
"A"—Completely or Practically Non-Magnetic (less than 1 per cent. Magnetism)			
300	168	0.3	215
400	6	0.2	340
400	12	0.6	340
450	6	0.2	337
530 Furnace	15	0.3	277
750 Water	1		
530 Furnace	15	trace	339
750 Water	1		
650	3	0.2	408
650	6	0.2	418
750	1	0.3	277
750	1	0.3	277
"B"—Partly Magnetic (1 to 12 per cent. Magnetism)			
300	552	1.25	217
300	384	1.0	317
300	384	1.0	328
400	24	1.0	364
400	48	2.0	387
450	12	1.2	361
450	24	2.2	387
450	48	7.0	218
600	3	12.2	444
"C"—Very Magnetic (20 per cent. Magnetism and over)			
500	6	20.4	351
500	12	28.8	351
500	24	34.4	444
500	48	40.2	438
500	600	42.0	444
550	6	20.0	468
550	12	24.8	400
550	24	25.8	438
550	48	31.4	495
600	12	19.8	418
600	24	24.4	375
600	48	30.0	418
600	72	29.8	430

sufficiently long heat treatment, there is no doubt this specific magnetism could be further and considerably increased.

One interesting point which comes out very clearly is that with the increase in specific magnetism, specially in the more magnetic material, the Brinell ball hardness numbers rise very remarkably; in fact these are nearly double as compared with those representing the original hardness of manganese steel, which is about 200.

In view of deductions which may be made from these experiments, it should be pointed out, and as shown by the tables, that it is quite possible to have manganese steel possessing a ball hardness number of even as high as 408, yet the material remains practically non-magnetic—namely, with only 0.20 per cent. of specific magnetism. It would hardly seem as if any support can be claimed by the allotropist from this fact, because it is also possible to have magnetic manganese steel of about the same ball hardness, yet with 30 to 40 per cent. specific magnetism.

In these specimens possessing high percentages of specific magnetism, the hardness in some cases reaches 495, but one of the specimens in group (C) shows a ball hardness number of 444, yet it possesses no less than 42 per cent. of specific magnetism. It might be interesting to state that glass-scratching hardness requires a Brinell hardness number of about 600 and upward.

The Allotropic Theory.—There may be those who will claim that the high specific magnetism noticed in such specimens may be due to the presence of alpha iron. Be that as it may, the writer considers that this cannot very well be reconciled with the fact that as the material rises in magnetic qualities, and assuming for a moment that this is due to the presence of alpha iron, why does the ball hardness rise to the high figures mentioned, in fact in some cases to over 500, which, as just mentioned, is rapidly approaching material having a quality of hardness which would scratch glass. It would be imagined that the presence of more alpha iron would make the material softer, whereas the exact contrary is the case; the higher the specific magnetism, the greater the hardness.

Effect of Treatment on the Magnetic Qualities of Nickel-Chromium Steel.—With regard to the quality of specific magnetism, as a comparison with manganese steel it may be mentioned that upon quenching other alloy steels, such as nickel-chromium steel containing say 0.65 per cent. carbon, this material in its annealed condition has a specific magnetism of 90 per cent., as compared with Swedish charcoal iron 100 per cent., and a ball hardness number of 212. Upon quenching this product in oil at a temperature of 940° C., the material has a ball hardness of 683, and its specific magnetism is reduced to 74 per cent. Table IV shows the different qualities in the (*a*) annealed, and (*b*) quenched conditions.

TABLE IV

Condition of Specimen	Ball Test	Specific Magnetism	Residual Magnetism	Coercive Force	Electrical Resistance
(*a*) Annealed.....	212	90	14,400	17.5	24.2
(*b*) Quenched in oil	683	74	7,800	43.7	41.9

It will be seen that on the material becoming hard, the specific magnetism has fallen about 20 per cent., whereas in the manganese steel referred to in this paper the specific magnetism increases, and at the same time the ball hardness number increases more than 100 per cent. In other words, as will be shown later on in the paper, these differences in magnetic qualities must, so the writer thinks, be ascribed to reasons altogether different from those offered by the allotropist. As the above figures show, remarkable changes have been produced by the treatments referred to, in physical, mechanical, and electrical qualities. The writer would urge these are due rather to changes in the form of the carbon combinations with the iron.

In the writer's paper read before this Institute in 1893, the following was stated: "It is also important by this experiment to have proved that the hardness of manganese steel is independent of its magnetic condition, confirming Mr. Wingham's contention several years ago (*Journal of Iron and Steel Institute*, 1890, No. I, p. 101) that the beta-iron theory is inconsistent with the facts noticed in magnetizing hardened steel. He rightly asked: How could permanent magnetism in hardened carbon-steel be reconciled with the fact that such material was found so exceedingly suitable for the manufacture of permanent magnets if the iron present in that material was in the beta-form, the latter being supposed to be non-magnetic? If hardened steel is a material in which non-magnetic beta-iron is imprisoned, so to speak, how comes it that such a steel is, of all others, the most suitable medium for articles requiring magnetic retentiveness? Mr. Wingham further added that to make the hypothesis more acceptable it would be necessary to say that 'apparently hardened steel must contain some alpha [or soft] iron.' This of course is quite inconsistent with the facts known. The harder the steel the more pronounced are its qualities of magnetic retentiveness. . . . Conversely, wrought-iron, in which there is supposed to be no iron present in the non-magnetic or beta-form, even if quenched from a high heat, ought to make a good permanent magnet, as all the iron present is supposed to be in the alpha and magnetic state. It is certainly subject to magnetic influence, but otherwise it has just the opposite quality."

The statements in the foregoing citation, while made by the writer so

long as 20 years ago, yet to-day seem to have quite as much force in them as when submitted to this Institute.

The allotropist has, so the writer understands, always claimed that when iron takes the beta form it is non-magnetic. In the specimens described in this paper, however, the same manganese steel which is before absolutely non-magnetic, after treatment possesses about 42 per cent. of the specific magnetism of pure iron, and the hardness, instead of decreasing, as it should do, that is if its latter qualities are due to the presence of alpha iron, rises to the respectable figure of nearly 500 Brinell ball hardness number. On the other hand, the iron and nickel alloy (C, 0.60; Mn, 1.00; Ni, 20.00 per cent.) which is non-magnetic at ordinary temperatures, remains non-magnetic even if heated for a considerable time, about 96 hr., to temperatures of between 300° and 700° C., yet this same material upon being cooled down in liquid air becomes magnetic. Moreover, its hardness, whether in the non-magnetic or magnetic condition, remains low, from 180 to about 250 ball hardness number.

The writer trusts that the deductions to be drawn from the experiments now described will prove of considerable practical and scientific importance. It is also further shown as a result of this research work, which commenced in 1894 when the writer read his paper already referred to before the Iron and Steel Institute, and now further continued during the last four years by Professor Hopkinson and himself, that by this treatment manganese steel can be brought into such a condition that it can be dealt with by tools.

The Relation of Hardness to the Magnetic Qualities.—It must not be overlooked that with regard to ordinary manganese steel, it is possible to have this material with say 200 ball hardness number, also with considerably over 500 ball hardness number, and in between these two conditions, yet in all these states or conditions the material is non-magnetic.

The writer has also made ball hardness tests on the half of a manganese-steel test bar which had been pulled in a tensile machine. The tensile strength was 62 tons per square inch, with an elongation of 52 per cent. on 8 in. The ball test on the unstressed end was 223, and near the fractured portion no less than 540.

While this hardness is produced by quite different treatment from the specimens described in the present paper, it is interesting to note that the hardness of manganese steel can be varied from the comparatively low figure of 223 up to 540, without in any way affecting its non-magnetic condition, even when the material is physically strained. As is well known, in ordinary steel the material often becomes a strong magnet after being physically strained. Manganese steel of the ordinary type, which is non-magnetic, on the other hand can be strained so that its hardness is more than doubled, without materially affecting its non-magnetic condition.

It may be interesting to set this out in the form of a diagram, Fig. 2. In this diagram comparison is drawn between manganese steel, non-magnetic nickel steel, and ordinary mild carbon steel. There is shown the actual increase in the ball hardness figures without regard to their

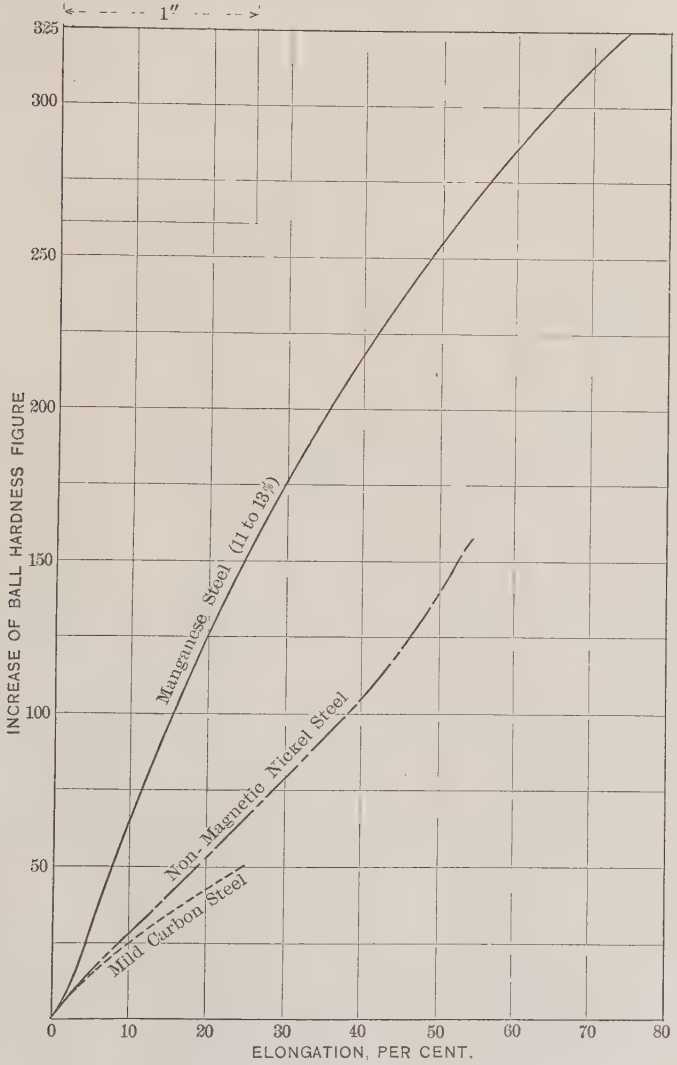


Fig. 2.—Increase of Ball Hardness by Stretching.

differences in initial hardness. It will be seen that if the specimen is stretched to an equal percentage of elongation, then the hardness number of manganese steel is increased much more than either carbon or nickel steel, nickel steel being intermediate. An elongation of 20 per cent.

produces in manganese steel an increase in ball hardness of 125; high nickel steel (22.8 per cent. nickel) increases only 50, and mild carbon steel 42. The actual ball hardness figures of these materials before stretching were: Manganese steel, 223; high nickel steel, 164; and mild carbon steel, 137.

It may be interesting to add that the same high nickel steel above mentioned (actual analysis, C, 0.59; Mn, 1.01; Ni, 22.80 per cent.) gave the following results in the tensile testing machine:

TABLE V

Test-bar number	6803F	6804
Treatment	As forged	800 abs. furnace
Tensile test:		
Elastic limit, tons	22	16
Breaking load, tons	48	43
Elongation, per cent	55	47
Reduction in area, per cent	72	58
Ball test:		
On thread	179	155
Near fracture	340	280
Magnetic properties, test with hand magnet:		
On thread	Non-magnetic	Non-magnetic
Near fracture	Slightly magnetic	Very feebly magnetic

It is quite clear, therefore, that non-magnetic nickel steel, like manganese steel, has its hardness increased by deformation. Unlike manganese steel, however, the high nickel steel is made distinctly magnetic by deformation.

Interesting results were also obtained by heating manganese steel, to a temperature of between about 500° and 550° C. as an approximate maximum. It is possible to then obtain manganese steel which is magnetic and possesses a ball hardness of about 300 to 350; this product can be tooled. By the further heat treatments described in this paper, the ball hardness of the material can be increased to about 500, with between 30 and 40 per cent. of specific magnetism.

The previous research work of the writer bears in an interesting manner upon the foregoing results. In his original paper of 1888 on Manganese Steel, read before the Institution of Civil Engineers, it was stated "Manganese steel of the lower percentages possesses the peculiar advantage of being softened by the following treatment: A specimen heated sufficiently to be seen red-hot in a dark place, on being plunged at this low heat into cold water, becomes sufficiently soft to be easily filed, planed, or otherwise fitted. Hardness is then restored by reheating to a bright red and cooling in air." Similar results were then reported by some of those joining in the discussion at that time.

The writer's attention has been called by his friend Knox Taylor to

the interesting and excellent laboratory practice book by Charles Morris Johnson, entitled *Rapid Methods for the Chemical Analysis of Special Steels, Steel-making Alloys and Graphite.* In the chapter on "The Annealing of Steel," Mr. Johnson pointed out more than five years ago that he had succeeded in softening manganese steel by heat treating it at a temperature of 520° to 550° C. He stated that by this treatment a plate could be drilled without sharpening the bit, also the plate was readily drilled without screeching. As an evidence of the correctness of the foregoing, four holes were drilled in manganese steel without resharpening the drill. Plates of the same material were also easily machined in the planer. It was noticed that some of the plates were also very brittle while in this condition.

Mr. Johnson also made some chemical determinations, and found, as the writer has pointed out in some of his previous papers, and as also noticed by Dr. Ledebur and one of his students at Freiberg, Tetskichi Mukai, of Tokyo, in the latter's paper, Studies on Chemico-analytical and Microscopical Examination of Manganese Steel, that the condition of the carbon in manganese steel can be largely changed by heat treatment. In the particular case before referred to, Mr. Johnson found that the carbon had changed into what he terms "almost entirely into the perfectly annealed state." The writer of this paper is heartily in agreement with Mr. Johnson's conclusions, as they help to bear out the view he has held for a long time, namely, that the condition of the carbon in manganese steel has an important bearing upon its physical quality. In other words, without seeking to explain the changes noticed by considering whether there has been an allotropic modification of the iron present, the changes in the carbon combinations in manganese steel may account for, or at any rate have an important bearing upon, the changes in its physical and magnetic qualities.

In the same direction, it would be interesting to add the results of the following experiment recently carried out on similar lines to those referred to in the writer's paper some 20 years ago. It was then, as now, noticed that it was possible to make manganese steel magnetic by the following treatment. This experiment is, however, more specific, with the exact temperatures recorded.

The specimens of manganese steel were heated as follows:

(a) In charcoal to 1,000° C. and cooled slowly in an electric furnace. This bar, while to some extent magnetic, is not nearly so much so as the material submitted to the standard treatment at 500° C. referred to in this paper.

(b) Another specimen was taken and heated in the same manner to 970° C., then cooled in furnace to 500° C., and kept for 69 hours at 500° C. This was very magnetic, fully as much so as any of the specimens referred to in the present research.

It would appear therefore that, as proved by the various experiments referred to in this paper, the mere heating of manganese steel for a long time in charcoal probably does not produce the same results as the specimens dealt with in this research. The heat treatments necessary to obtain the full effect are those referred to in the present paper, and as mentioned by Mr. Johnson in the above-quoted reference from his work. In other words, the temperatures quoted by Mr. Johnson appear to be practically the same as those by which Professor Hopkinson and the writer have obtained in the various specimens treated the many curious magnetic and physical changes described in this paper.

The results of these various experiments open a wide field of scientific speculation, and the writer cannot but think the results described in the present paper will eventually be found to help those scientists who are studying the properties or qualities of what are termed magnetism or magnetic forces.

The Excellent Work of Professor Pierre Weiss on This Subject.—For example, the results of the interesting research work by Prof. Pierre Weiss, on The Magnetic Moment of Atoms and the Magneton, described in his paper to La Société Française de Physique, Paris, Jan. 5, 1913, appear to show that this quality of specific magnetism of the metals in the iron group or their alloys, probably depends upon quite other factors than allotropic modifications of the elements present. Professor Weiss claims that in what he terms "Magnetons" he has discovered better methods of explaining magnetic phenomena than by other theories. He appears to consider that the increase of magnetons, with their important bearing upon magnetic qualities, is independent of allotropic or perhaps even chemical modifications.

Moreover, Weiss, to use his own words, and to quote from him fully on this important subject, has made the following statement. "In 1905 Kamerlingh Onnes and myself undertook the determination of the atomic moments of iron, nickel and cobalt, and the molecular moment of magnetite, Fe_3O_4. For this purpose we compared the magnetization to saturation of these substances in liquid hydrogen, which boils at 20° absolute scale, at atmospheric pressure, with the known value of this quantity at the ordinary temperature. For iron and nickel the experiment succeeded, and the appearance of the magnetization curves showed that saturation had been reached. For cobalt and magnetite this state was not reached, due doubtless to the potential energy of magneto-crystalline origin preventing complete alignment. We found for the moment of the gram-atom:

Iron	12.360
Nickel	3.370

We have also investigated the question of the rationality of the relations between the atomic moments, and have reached a negative conclusion.

If we had made the comparison with definite numbers, instead of making it with roughly estimated numbers, before making various slight corrections, we would perhaps have been struck with the extreme exactness with which the ratio 11:3 is obtained. We have in fact:

Iron..... 12.360=11×1123 6
Nickel..... 3.370= 3×1123.3

I shall here anticipate the demonstration which I shall give of the generality of a rational relationship between the atomic moments and shall call the common aliquot fraction 1123.5 the *Magneton.* As the numbers here relate to the gram-atom, 1123.5 will be the magneton-gram. By dividing by the number of Avogadro 68.5×10^{22}, we obtain the magneton itself, that is, $16.4\times10\text{-}^{22}$. This is the moment of a small elementary magnet which will be contained eleven times in the atom of iron, and three times in the atom of nickel. The magnetons of the same atom will be exactly parallel to each other. For cobalt the gap was later filled indirectly by a series of measurements which I asked O. Bloch to make with the alloys of nickel and cobalt (Bloch, thesis, Zürich, 1912). Up to 70 per cent. of nickel we obtain without trouble, saturation at low temperatures, and this quantity varies in a direct line with the percentages. It follows the law of mixtures. In prolonging the line up to 100 percentage of cobalt, we find 8.925 magnetons, that is nine according to the degree of accuracy of the experiment."

He then goes on to define the magnetic moment of a magnet as "the product of the magnitude of one of its poles and their distance; when a body contains several parallel magnets, the resulting magnetic moment is the sum of the magnetic moments composing it."

He further points out that, with regard to the properties of the alloys of the ferro-magnetic alloys, "Many interesting points have been revealed. In solid solutions magnetization to saturation in the neighborhood of the absolute zero varies lineally as a function of the centesimal composition. The chemical compounds have, on the contrary, their number of the magneton proper. In the ferro-nickel series, the compound Fe_2Ni, whose existence, it may be said in passing, has been really revealed by the magnetic properties, possesses for the entire molecule Fe_2Ni, thirty magnetons. Magnetization to saturation at very low temperatures is represented within the space of 0 to 100 percentage of nickel by two lines, one of which extends from Fe to Fe_2Ni, and the other from Fe_2Ni to Ni."

Weiss also points out that "While we know as yet but little regarding the properties of the magneton, yet the nature of the change or modification found in bodies whose number of magnetons varies with a determined temperature, sometimes with great suddenness, thus making us think of a point of fusion or melting point, appears to be very mys-

terious and requires in itself special study." He adds "This is not an allotropic transformation in the ordinary sense of the word, since in the case of magnetite (Fe_3O_4) this body shows that the molecule preserves throughout these transformations its general construction with the same number of degrees of freedom. Does this modification require the expenditure of energy? Are there other external characteristics which accompany it? It is possible to give an answer to this last question on one point only. The constant of the molecular field, which expresses the mutual action of direction in the ferro-magnetic bodies, suffers a change in common with the number of magnetons. The correlation between the number of magnetons in the same atom of iron, for example, and the chemical type of compounds in which it figures, deserves to be studied very carefully. Are the chemical forces elementary magnetic attractions? Are the valences in either case similar to magnetons?"

Weiss concludes his paper by stating: "Finally, as we may say, that as the electron shows the new ideas on the irregular and discontinuous structure of electricity, so the magneton marks as analogous evolution in the representation of magnetic phenomena."

As regards Weiss's work, this is of special importance, as his theories show that it is possible to definitely account for and explain the physical changes produced in the so-called alpha-beta transformation, A_2, or in other words, the change in the magnetic properties and specific heat of iron at the critical point in question. These changes are explained by the mechanism of the sub-atomic masses, which are termed magnetons by Weiss. In view, therefore, of his work, it would appear that it is not necessary to look to allotropic theories for an explanation of the phenomena in question.

What is the Explanation of the Peculiar Facts Noticed?—Coming as these theories do from such an able worker as Professor Weiss, they must in the near future prove of great importance. The writer, however, would point out that chemical modifications or combinations of the constituents of ferro alloys also offer a field in which at any rate a partial explanation seems to be obtained of some of the many peculiar facts noticed in connection with the research now described. For example, the writer thinks that the behavior of carbon in its various forms, as carbides of iron, or carbides of manganese, or both, their dissociation, their combination, probably have more or less important bearing on the many peculiar magnetic changes noticed in the manganese-steel specimens now described.

Elmer H. Williams, of the University of Illinois, in a very interesting paper on The Electron Theory of Magnetism,[1] states that some bodies,

[1] Mr. Williams points out to illustrate the size of an electron as compared with an atom, that one must imagine a hydrogen atom increased in volume to that of a large cathedral, the electron being increased proportionally. Then the volume of the electron would be that of a fly flying about in the vast space.

such as iron, when heated, show a gradual transition from the ferro-magnetic to the paramagnetic state, or *vice versa*, but as yet no body, with the exception of tin, has been found which, by change of physical conditions, will pass from the diamagnetic to the paramagnetic state.

It would be interesting to further follow up Mr. Williams's researches as so ably set forth in the paper referred to above.

With regard to some of the foregoing points submitted for consideration, it is also to be noticed that Honda and Takagi, in their recent and most valuable paper upon The Transformation Points of Iron and Steel at High Temperatures (*Revue de Métallurgie*, December, 1913), point out that the disengagement of heat in such transformations is not noticeable, and still more important, they mention that recalescence cannot be attributed, as believed by some, to the polymorphous transformations of the iron, but that it is due to the disengagement of heat produced by the separation of the mixed crystals of iron.

P. Goerens in his paper on The Influence of Thermal Treatment upon the Properties of Strained (écroui) Steel (*Revue de Métallurgie*, December, 1913) has pointed out that in experiments which he carried out upon strained steel, while annealing will restore the original properties of such material, this does not occur so long as the temperature of annealing remains below 520° C. He found this temperature, 520° C., was constant not only for one but various kinds of steel, the temperature of such change varying between 520° and 560° C. This is not a little remarkable seeing that these figures represent about the same temperatures at which the various manganese-steel specimens referred to in this paper become considerably magnetic; for example, the specimens represented by group (C) in Table I.

It must not be overlooked that some of the thermal change points in iron are no doubt due to changes in the specific heat qualities. Weiss points out that the irregularities in the specific heats of ferro-magnetic bodies present veritable riddles, though he considers that, by a proper study and comparison of the magnetic and calorimetric results, a solution appears to be possible.

While fully recognizing how exceedingly valuable free and open discussion on the theories of the allotropist and the carbonist has proved, it would be a great boon if some better understanding could be arrived at as to the meaning of the facts noticed with regard to magnetic phenomena in connection with the alloys of iron and steel. It is admitted that they are very complex, but in the light of the enormous amount of research work which has been carried out during the last few years, it would seem that there is an opportunity for the various schools of thought to meet on some common ground, with a view to work together in that direction which seems to offer the most light, and thus enable us to understand and eventually apply such benefits as are obtained to practical advantage.

Concluding Remarks.—In conclusion, it is hoped that the peculiar behavior now described of manganese steel when subjected to heat treatment, may throw some light upon other phenomena met with in alloys of iron with various elements, including carbon steel, which can also be included as an alloy steel. The experiments have been carried out as carefully as possible, and it has taken several years to deal with and try to unravel some of the peculiar results obtained.

The writer would also here take the opportunity of thanking the willing workers in his own Research Laboratory at Sheffield, Messrs. Milne and Main, for their co-operation and assistance.

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Discussion

W. S. Potter, Pittsburgh, Pa.—The writer of this paper speaks of heating specimens of rolled manganese steel containing carbon 1.21 per cent., manganese 12.36 per cent., in a cementation furnace, and of obtaining various degrees of magnetic increase in the different specimens.

Under the heading "A," completely or practically non-magnetic, the writer mentions specimens Nos. 5972 and 5973, each heated 15 hr. at 530° C., while under the heading "C," very magnetic, he lists specimens 5748 and 5750, heated at 550° and 500° C., each for 6 hr. (in all four cases the metal was cooled with the furnace). A lower magnetic effect was observed in the one case after 15 hr. heating at 530° C. than in the other two specimens heated for the lesser time, one at 550° C. and the other at 500° C.

I would ask the writer what analyses, particularly as to carbon, were found in the various specimens in the lots "A," "B," and "C" after the heat treatment. In July, 1911, about two carloads of manganese-steel sheets were rolled at the works of the American Sheet & Tin Plate Co. at Vandergrift, Pa. As an experiment, one sheet was annealed in the box annealing furnace. This sheet was hard, brittle, and highly magnetic. It went into the furnace with carbon 1.17 per cent. It came out with a carbon content of 0.65 per cent.

In my paper presented to this Institute at this meeting on the subject of manganese steel, the various heat treatments as per Series 2, with the exception of heat treatments 1, 2, and 3, caused the bars to suffer a very notable change in respect to magnetic qualities; that is to say, manganese-steel bars heat treated for 1 hr. at temperatures of from 371°

C. to 1,204° C. were much more magnetic than in the original rolled and water-quenched state of the steel, but this was confined to the skin of the bars, except as follows: A magnetic increase was just noticeable in the interior portions of the bars heated to 400° C. to 700° C., but not in those heated to higher temperatures. The sorbitic and martensitic conditions of the steel have perhaps a slightly greater magnetic quality than the austenite state. The strong, unmistakable magnetic effects, so far as I have observed them, are, however, accompanied with loss of carbon, and a general break up, which might be expected to liberate iron from a double carbide of iron and manganese.

I repeat my question, What were the analyses of Sir Robert Hadfield's specimens after the various long heatings which he mentions?

Albert Sauveur, Cambridge, Mass.—This paper has interested me very much, but apparently I am not looking at the question from the standpoint of the authors, because while they show in their paper a very strong argument against the allotropic theory, I find it one of the best supports of the allotropic theory ever offered. In my opinion, the behavior of manganese steel under heat treatment can be accounted for on terms of the allotropic theory, and, so far as I can see, in no other way.

Henry M. Howe, New York, N. Y.—I think the Chairman has spoken very justly in saying that the experiments of Sir Robert Hadfield are a very strong support, not of the allotropic theory—that does not require any support, that is as firmly established as the dynamic theory of heat compared with the phlogistic theory of heat. I do not think it is open to discussion. Phlogistic theory of heat is a little longer dead. The theory that we have no allotropy in iron is equally dead with the phlogistic theory. It simply is not quite so long dead.

The case of manganese steel is greatly complicated from the fact that from it the cementite precipitates with great readiness, and that makes the phenomena much more difficult to interpret, but this evidence that you pass to the magnetic state so readily, it seems to me, is fully in accord with the idea which I expressed this morning earlier, that you have this beta state intermediate between the gamma and the alpha state. In the case of nickel steel, to which I referred, that is now to a large extent paralleled by that of manganese steel, that you could, by proper heat treatment, transfer from the gamma state to this magnetic state.

B. Hopkinson, Cambridge, England (communication to the Secretary*).—I am very glad that Sir Robert Hadfield has been able to present a paper to the American Institute of Mining Engineers on manganese steel and to include in it some account of the research on which we have recently been engaged together. I need hardly say that I entirely agree with the statement which he has given of the general results of our joint

*Received Mar. 7, 1914.

work. I should like to add one point which has arisen since Sir Robert Hadfield left this country for the United States. He refers in his paper to the hardening or stiffening of water-toughened manganese steel which occurs when a test piece is pulled out and broken; the parts near the fracture have 2 or $2\frac{1}{2}$ times the Brinell hardness number that the metal possessed before the fracture. An accurate magnetic test which I have made recently on one such bar which Sir Robert Hadfield sent to me just before he left shows that the hardened portion near the fracture is very slightly, but quite distinctly, magnetic. The magnetism in *intense* fields is about 0.8 per cent. of that of pure iron,[2] whereas a piece cut from the unstrained ends of the same bar would show less than 0.3 per cent. This fact does not, of course, affect Sir Robert Hadfield's general conclusion that profound changes in mechanical properties may be associated with very slight changes in magnetic properties and presumably therefore with but little change in the molecular condition (whether by the formation of allotropic modifications or otherwise) of the iron or other elements present.

The fact that manganese steel which has been hardened by pulling is, though only slightly magnetic, still many times as magnetic as the same steel before the work has been done on it, has some significance, however. Taken together with much other evidence which cannot be summarized here but which will be given in detail in the paper which Sir Robert Hadfield and I presented last October to the Iron and Steel Institute, to be read in May, it seems to me to show that the hardness of this material when in the magnetic condition is due to the presence of a hard constituent, which is produced by heating the water-toughened steel to any temperature between 400° and 600° C. or by mechanical work. As the amount of the constituent is increased the magnetism increases with it, but whether this is because it is itself magnetic or because the remainder is made magnetic by the abstraction of carbon or manganese into the new constituent, the evidence is as yet insufficient to decide. Stead, who proved the existence of such a constituent in 1894, was of the latter opinion. It is apparently formed as a continuous network inclosing separate grains of (presumably) still soft metal. In consequence of this distribution even small quantities of it such as are associated with trifling changes of magnetism can greatly affect the mechanical properties of the material, just as a trace of lead added to gold entirely destroys its ductility by forming a brittle eutectic inclosing the grains of pure metal. Facts such as these suggest that no invariable relation ought to be expected between the mechanical properties of a steel and the molecular condition of its constituents, because so much depends upon the arrangement of these constituents. The production by heat treatment of a hard constituent (say the beta iron of the

[2] In ordinary fields it, of course, remains practically non-magnetic.

allotropists, assuming such to exist), even in considerable quantities, might leave the material perfectly ductile if it were formed in separate nodules inclosed in a matrix of ductile metal. On the other hand, quite a small quantity of the same material arranged as a continuous network might confer hardness. It seems to me that any theory which fails to take full account of this important factor of arrangement is likely to lead to wrong conclusions.

I ought perhaps to add that the views expressed in the last paragraph must be taken as the expression of my own opinion only, as I have not yet had the opportunity of discussing this interpretation of the facts fully with Sir Robert Hadfield.

Sir Robert Hadfield (communication to the Secretary*):—I note with much interest Mr. Potter's remarks regarding the decarbonization of a manganese-steel sheet. A thin sheet may be readily decarbonized, so that a reduction from 1.17 to 0.65 per cent. carbon might be expected. In our own practice in Sheffield we have had similar experience.

As regards the latter portion of Mr. Potter's remarks, he says that "manganese-steel bars heat-treated for 1 hr. at temperatures of from 371° C. to 1,204° C. were much more magnetic than in the original rolled and water-quenched state of the steel." If by this he means that upon heating to 371° C. the material becomes magnetic, then it is quite certain he would not be able to obtain magnetic qualities. The temperatures referred to in our paper must be reached before this change from non-magnetic to magnetic condition occurs. It was one of the objects of presenting this paper to identify and make clear the exact temperatures, or as near as is possible, at which this extraordinary change occurs; that is, when manganese steel passes from the ordinarily non-magnetic into the magnetic condition.

With regard to the various specimens referred to in our paper, and the question asked by Mr. Potter, I can assure him most positively that there was practically no alteration in the percentages of carbon, whether by short, medium, or long heat treatment. We have repeatedly tested and proved this. If Mr. Potter wishes, I would be very pleased to send him specimens of our manganese steel before and after the treatment referred to in the paper, so that he himself could verify the statement.

I would again state most positively that the magnetic effect produced has been obtained entirely apart from any loss of carbon in the steel. Mr. Potter speaks of "a general break up." This term seems somewhat vague, and it is difficult to answer his question without more definite information as to what he has in mind by the use of this term.

I note with interest Professor Sauveur's remarks, and in reply would say that as I have dealt with the points with regard to allotropy in my

*Received April 13, 1914.

discussion on his paper entitled Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, it would be better to refer the reader to this for fuller comments.

In the same way as regards Professor Howe's remarks. I do not, however, quite understand what Professor Howe means by the allotropic theory, which in the language of the founder (M. Osmond) was: "I conclude that hardened steel owes its properties principally to the presence of beta iron, which is hard and brittle by itself at the ordinary temperature." If Professor Howe includes this as one of the basic points and claims that there is a hard beta adamantine form of iron, then not only is this seriously open to discussion, but such theory has now been proved to be untenable. Surely it is the allotropic theory, if it is intended in this sense, which is dead.

I would suggest that those interested in this matter, before making any further positive statements with regard to the allotropic theory, should read the recent important papers by A. McCance on The Theory of Hardening, Professors Edwards and Carpenter on The Hardening of Metals, and Professor Honda on Transformations of Iron and Steel at High Temperatures. These three authors clearly prove that the much-disputed A2 point represents no allotropic change at all; also that beta iron does not exist.

I quite agree with Professor Howe's remarks regarding cementite being precipitated with great readiness in manganese steel.

Manganese Steel and the Allotropic Theory

BY ALBERT SAUVEUR, CAMBRIDGE, MASS.

(Pittsburgh Meeting, October, 1914)

At the New York meeting of the Institute, February, 1914, Professor Hopkinson and Sir Robert Hadfield presented an important paper entitled Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel.[1]

Any contribution on manganese steel by its discoverer necessarily carries with it much weight and is entitled to serious and close consideration. The momentous discovery of that alloy by Sir Robert Hadfield some 30 years ago was the signal for great activity in the field of alloy steels; and the remarkable results obtained in the last two decades probably constitute the most important modern advance in the metallurgy of steel.

The authors of the paper referred to above are unable to reconcile with the allotropic theory the interesting results obtained by them in heat treating manganese steel. When they speak of the allotropic theory, however, I understand that they have more especially in mind the existence of beta iron. It is not my understanding that they question the occurrence of iron in the two allotropic varieties known as gamma and alpha. If I am wrong, I trust they will correct me.

A careful consideration of the experimental results reported and discussed by the authors leads me to believe that on the contrary they can be very satisfactorily explained in terms of allotropy and that, in the light of our present knowledge, they cannot be accounted for in any other way.

The physical properties of non-magnetic manganese steel described by the authors are those pertaining to iron-carbon alloys in which iron is present in the gamma condition, or, in other words, to austenite. The metal is then characterized by great tenacity and ductility and low elastic limit, while its hardness, mineralogically speaking, is not excessive, but of a special kind termed "tough hardness" by the authors. Even when subjected to the restraining influence of a large amount of manganese and of considerable carbon, however, relatively rapid cooling is necessary to produce austenitic manganese steel free from beta and

[1] This volume, p. 476.

alpha iron. On very slow cooling from a high temperature, as indicated by the authors, manganese steel becomes, to some extent, magnetic, because in accordance with the allotropic theory such slow cooling permits the formation of some beta and alpha iron. The mineralogical hardness of the metal is then, as it should be, somewhat greater. Non-magnetic, and, therefore, austenitic, manganese steel, while very much more stable than hardened, high-carbon steel, is nevertheless in an unstable condition, as evidenced by the fact that relatively quick cooling is required for its production. On reheating (shall we say tempering?) manganese steel, therefore, we may logically anticipate a partial transformation of gamma iron into beta and, possibly, into alpha iron—and, indeed, the authors show that reheating non-magnetic, austenitic manganese steel for a long period of time to some 500° C. or higher causes the return of considerable magnetism and greatly increased hardness. While they consider this occurrence fatal to the allotropic theory, it appears to me, on the contrary, to carry with it a striking confirmation of the correctness of that theory. If there were no hard allotropic modifications of iron *between* the soft alpha and moderately hard gamma varieties, the transformation taking place in reheating manganese steel could only imply the formation of some alpha iron and, therefore, *decreased* hardness. The fact that the treated steel becomes magnetic also points to the formation of some alpha, together with beta iron. This is also consistent with the allotropic transformation observed in carbon steel when we find that whenever a notable quantity of beta iron is formed, causing hardness, some alpha iron also forms, imparting magnetism to the metal; hence the hardness and magnetic properties of hardened carbon steel. It does not seem possible, in the presence of carbon at least, to produce much beta iron to the exclusion of alpha iron. These two allotropic varieties probably form isomorphous crystals which are both hard and magnetic.

The authors seem to have some difficulty in explaining, in accordance with the allotropic theory, the fact that on heat treating non-magnetic manganese steel, both the hardness and the specific magnetism may *increase*, their argument being that if magnetism is due to the formation of soft alpha iron, then increased magnetism must mean increased softness. This, however, is only an apparent inconsistency, as I have attempted to show graphically in Fig. 1. We may assume that at 400° C. originally non-magnetic manganese steel remains non-magnetic, the iron present being wholly in the gamma condition. After a long heating, say at 450° C., some of the gamma iron is converted into beta and into alpha iron. Of the iron present, AB per cent. may be assumed to occur now as alpha iron, BC per cent. as beta iron, and CD per cent. as gamma iron. Heating to a higher temperature, or possibly a longer heating at the same temperature, causes further transformation, *both* beta and alpha iron *increasing*, the

metal containing now $A'B'$ per cent. of alpha and $B'C'$ per cent. of beta iron; hence its increased magnetism and increased hardness.

Some beta and alpha iron having now been formed, it is evident that on heating the metal to a sufficiently high temperature, a critical point must necessarily be found, caused by the return of alpha and beta iron to the gamma condition—the only one stable at a high temperature. The authors have located this point at about 675° C., as shown in my Fig. 1. On again cooling the metal relatively quickly, no critical point should be detected, since the steel remains non-magnetic, that is, in the gamma condition.

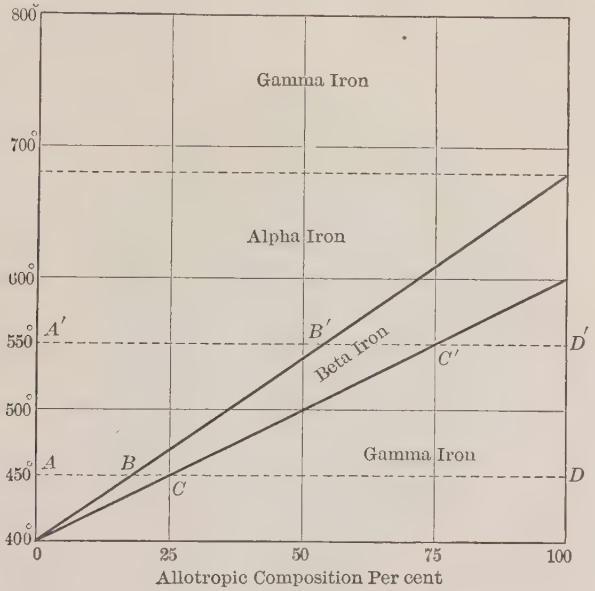

FIG. 1.

The fact alluded to by the authors that manganese steel can be so softened by suitable heat treatment as to make it machineable is another evidence of the formation of alpha iron.

From the fact that cast manganese steel is much less ductile than the water-toughened metal, although likewise non-magnetic, we not illogically infer that some beta iron may have been formed during the relatively slow cooling of the casting, while the usual lack of ductility of steel castings in general and the possible separation of carbide may be additional reasons for the properties of cast, untreated manganese steel.

The fact that manganese steel does not become magnetic on cooling it in liquid air, while non-magnetic nickel steel with some 22 per cent. nickel regains its magnetism after such treatment, does not point to

different influences being at work. It can be satisfactorily accounted for, as shown in Fig. 2, in which it is reasonably assumed that with 1 to 1.25 per cent. carbon it requires some 7 per cent. of manganese to lower the critical point from its normal position (some 700° C.) to atmospheric temperature, whereas some 18 per cent. nickel may be required to cause the same depression. Extending below atmospheric temperature the lines indicating the lowering of the critical points, it is seen that with some 12 per cent. manganese the critical transformation may be lowered to some −400° C., while nickel steel with 22 per cent. nickel would have a critical point at −100° C. and therefore detectable on immersion in liquid air. It will, of course, be understood that no claim is made here in regard to the exact position of the lines or curves indicating the

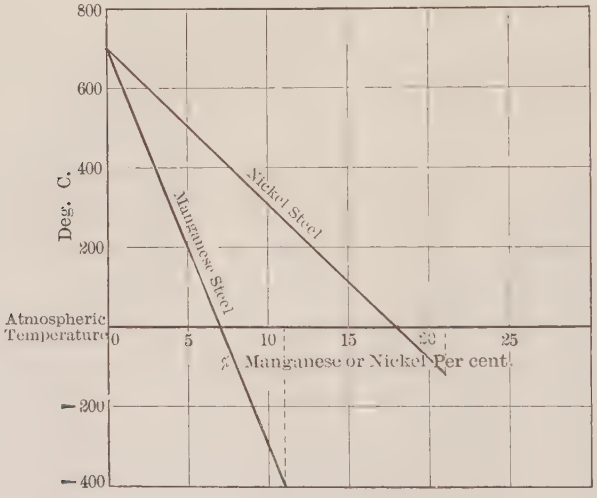

Fig. 2.

depression of the magnetic change with increased manganese or nickel. These positions, so far as I know, have never been accurately determined, but, from the results available, the diagram of Fig. 2 is not altogether a fantastic one.

The authors quote at length from the important work of Pierre Weiss, who has developed a new theory of magnetic transformation in metals.

The attitude of a scientific man toward new theories should be one of open-mindedness and receptiveness—even of reverence—when the new thought springs from an exalted source; but it should also be one of caution. He should guard against too hasty and enthusiastic acceptance. He should be mindful of the theoretical scrap heaps lining both sides of the pathway of scientific progress. For a while, at least, a new theory must be placed on probation, while being subjected to searching tests. It is the present position of Professor Weiss's theory.

With a view to securing, if possible, additional information in regard to the relation between the heat treatment of manganese steel, and its critical points, structure and physical properties, some experiments were conducted in the Metallographical Laboratory of Harvard University which will be briefly described and illustrated.

Steel Used

The manganese steel used in these experiments was supplied by the Taylor-Wharton Iron & Steel Co. of High Bridge, N. J., in the shape of cast bars $\frac{1}{2}$ by $\frac{3}{4}$ in. in cross section and about 12 in. long. The steel contained 1.25 per cent. carbon and about 12.50 per cent. manganese.

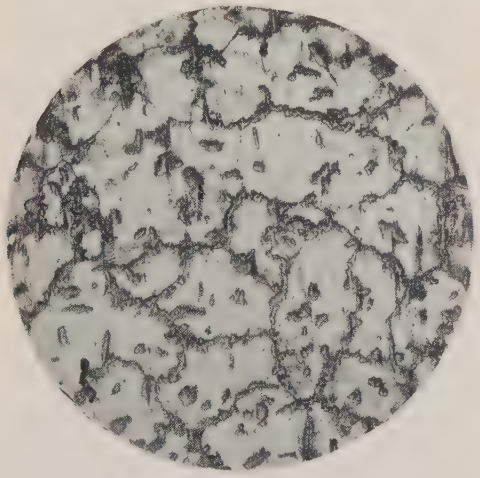

FIG. 3.—CAST MANGANESE STEEL. MAGNIFIED 100 DIAMETERS. CEMENTITO-AUSTENITIC STRUCTURE.

Cast Untreated Manganese Steel.—The structure of the cast metal is illustrated in Fig. 3. It consists of polyhedric grains of austenite surrounded by a considerable amount of a constituent generally held to be a double carbide of iron and manganese (manganiferous cementite), or a mixture (possibly a solid solution) of the carbide of iron Fe_3C and the carbide of manganese, Mn_3C. As is well known manganese steel in this condition has a low tensile strength, low elastic limit and little ductility. It is non-magnetic. Its Brinell hardness number[2] was found to be 196. In the light of the allotropic theory the absence of ferro-magnetism is to be

[2] The hardness tests mentioned in this paper were made at the Watertown Arsenal, Watertown, Mass. They will be found tabulated at the end of the paper.

ascribed to the absence of alpha iron while the lack of ductility of the metal may be accounted for through the formation during the relatively slow cooling of the castings of some beta iron, although, undoubtedly, the presence of segregated carbide surrounding the austenite grains and the usual structural coarseness of cast steel may be also regarded as causes of brittleness.

Heat Treatments

Some of the steel bars were subjected to the following treatments:

Heating to 1,100° C. and quenching in water.
Heating to 1,100° C. and cooling in furnace.
Heating to 1,100° C. and quenching in water followed by:
Heating to 800° for 2 hr. and cooling in furnace,
Heating to 700° for 2 hr. and cooling in furnace,
Heating to 600° for 2 hr. and cooling in furnace,
Heating to 500° for 2 hr. and cooling in furnace,
Heating to 400° for 2 hr. and cooling in furnace,
Heating to 575° for 90 hr. and cooling in furnace.

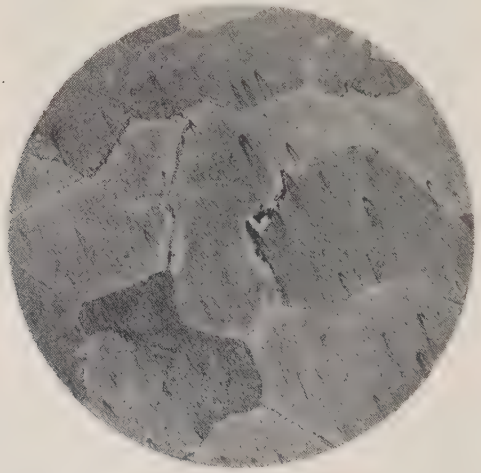

FIG. 4.—MANGANESE STEEL HEATED TO 1,100° C. AND QUENCHED IN WATER. MAGNIFIED 100 DIAMETERS. AUSTENITIC STRUCTURE.

Before subjecting the treated specimens to microscopical examination and to the Brinell test for hardness the outside, decarburized portion was removed by grinding.

Bar Heated to 1,100° *C. and Quenched in Water.*—The structure of this bar is illustrated in Fig. 4. The steel is now made up of polyhedric grains of austenite, quenching from a high temperature having caused the disappearance of the segregated carbide so conspicuous in the struc-

ture of the cast metal (Fig. 3). We naturally infer that at a high temperature this carbide or mixture of carbides dissolves in the austenite and that rapid cooling does not permit its re-precipitation. To the absence of this carbide must be ascribed in part at least the much greater toughness and tenacity of the quenched metal, although it may reasonably be claimed that the marked change of properties may also be due to the now complete absence of beta iron. As is always the case with austenitic steel, the elastic limit remains low. Two bars subjected to water quenching from 1,100° were tested for hardness, giving respectively as factors

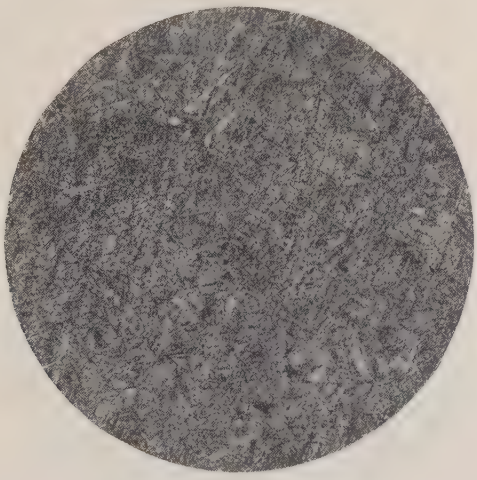

FIG. 5.—MANGANESE STEEL HEATED TO 1,100° C., QUENCHED IN WATER, REHEATED AT 700° FOR 2 HR. AND SLOWLY COOLED IN FURNACE. MAGNIFIED 100 DIAMETERS. AUSTENITO-MARTENSITIC STRUCTURE.

of hardness 174 and 181. This slightly decreased mineralogical hardness as compared to the hardness of the untreated metal (196) is in harmony with the conception of the formation of some beta iron during the relatively slow cooling of the cast bars and complete absence of it in the quenched bars. The quenched samples because of the absence of alpha iron are of course non-magnetic.

Bar Heated to 1,100° *and Slowly Cooled in the Furnace.*—The structure of this bar was similar to the structure of the cast untreated metal (Fig. 3). This was to be anticipated. Heating to 1,100° caused the segregated carbide or carbides to dissolve in the austenite but the very slow cooling that followed resulted in its being again precipitated. The metal was brittle and lacking in tenacity. Its hardness number was found to be 212, a slight increase over the hardness of the untreated metal (196). This may be explained on the ground that the furnace-

cooled bar cooled more slowly than the cast bars, resulting in the formation of a greater amount of beta iron. From the fact that this bar is non-magnetic we infer that although slow, the cooling was not slow enough to permit the formation of any appreciable quantity of alpha iron.

Bar Heated to 800° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar was quite similar to that of the cast, untreated bar, but from its greater hardness, which is now 245 as compared to 196, we infer that the long exposure to 800° followed by slow cooling made possible the formation of a greater amount of beta iron. The bar, however, is still non-magnetic, which points to the non-formation of any appreciable quantity of alpha iron.

FIG. 6.—SAME AS FIG. 5. MAGNIFIED 500 DIAMETERS.

Bar Heated to 700° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar, illustrated in Figs. 5 and 6 under two different magnifications, clearly indicates the formation of martensite and hence of considerable beta iron. From our knowledge of the properties of martensite we shall expect the steel to be ferro-magnetic because of the usual occurrence of an appreciable amount of alpha iron in martensite. We shall also expect the steel to be now mineralogically harder than when in an austenitic condition. These expectations are fulfilled. Manganese steel so treated is very appreciably ferro-magnetic and its hardness is now 266, or 70 points higher than in the cast condition and some 90 points higher than after quenching from 1,100°.

Bar Heated to 600° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar was similar to that of the preceding bar, that is, decidedly martensitic, while the bar, is ferro-magnetic, from which it

follows that the heat treatment to which it was subjected resulted in the production of beta and alpha iron. Its hardness number, 335, indicates a very great increase of hardness. Evidently this bar must contain more beta iron than the bar heated for 2 hr. to 700°. This is readily explained on the ground that 700° is very near the range of temperature in which gamma iron is the stable condition.

Bar Heated to 500° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar was also found to be martensitic while the metal was ferro-magnetic, but to a decidedly less degree than the preceding bar. Clearly, at 500° tempering does not proceed as far as it does at 600, with the result that less beta iron and less alpha iron are formed. This is perfectly consistent with the allotropic theory. The hardness number of the bar was 255.

It seems evident that a temperature of 600° or thereabout is the most effective one to render manganese steel martensitic and, therefore, hard and ferro-magnetic.

Bar Heated to 400° *and Slowly Cooled in the Furnace.*—The structure of this bar was practically that of the untreated quenched metal. It was non-magnetic and its hardness number was 163. Clearly, heating to 400° for 2 hr. is not sufficient to cause any appreciable tempering of the quenched metal. There is no apparent explanation for the fact that the steel is now somewhat softer than in the quenched condition.

Bar Heated to 575° *for* 90 *hr. and Slowly Cooled in the Furnace.*—Seeing that magnetism is readily produced by heating in the vicinity of 600°, while the structure of the metal becomes decidedly martensitic, a quenched bar was kept for 90 hr. at a temperature of 575°. The structure of this bar was of the type illustrated in Figs. 5 and 6; its hardness was 356 and the metal was so magnetic that the bar could be readily picked up by a permanent magnet.

Conclusions

From the above data it appears that manganese steel of the Hadfield type may occur under three distinct conditions: (1) as austenitic steel mixed with a considerable amount of free carbide, a condition which is produced by slow cooling from a temperature exceeding 700°; (2) as austenitic steel practically free from segregated carbides, a condition produced by rapid cooling from a high temperature; and (3) as martensitic or possibly austenito-martensitic steel, a condition most readily produced by reheating austenitic steel for a sufficient length of time to a temperature exceeding 500° but not 700°.

In the first condition, which may be described as cementito-austenitic, the metal is non-magnetic, weak and lacking in ductility, while its hardness number is in the vicinity of 200. In its austenitic condition

Brinell Hardness Tests

Marked (Treatment)	Readings, mm.	Average diameter mm.	Hardness Factor
Untreated unmarked.....	4.30 4.30 4.30	4.30	196
1100W.....	4.40 4.50 4.55	4.48	181
1100W duplicate.....	4.55 4.55 4.55	4.55	174
1100F	4.15 4.15 4.15	4.15	212
800–2–F.....	3.90 3.85 3.85	3.87	245
700–2–F.....	3.70 3.70 3.75	3.72	266
600–2–F.....	3.35 3.30 3.35	3.33	335
500–2–F.....	3.80 3.80 3.80	3.80	255
400–2–F.....	4.70 4.70 4.70	4.70	163
575–90–F.....	3.20 3.25 3.25	3.23	356

All tests were made under load of 3,000 kg. with 10-mm. ball pressure held for 30 sec. 1100W signifies heated to 1,100° C. and quenched in water; 800-2-F signifies heated to 800° for 2 hr. and furnace-cooled, etc.

the metal has a high tenacity, great ductility, is non-magnetic, and its hardness number is in the vicinity of 180.[3]

The martensitic condition results from the tempering of austenite, as previously explained. The metal is now ferro-magnetic because of the presence of some alpha iron, while its hardness varies between 250° and 350,[4] an increase due to the formation of a considerable amount of beta iron.

Discussion

G. K. Burgess, Washington, D. C.—This very instructive contribution by Professor Sauveur to our knowledge of that puzzling substance, Hadfield's manganese steel, should not only be read in connection with the paper of Messrs. Hopkinson and Hadfield referred to by Professor Sauveur; but one should also have in mind, as throwing much additional light upon the questions raised, two papers recently presented before the Iron and Steel Institute, the one by Sir Robert Hadfield (*Journal of the Iron and Steel Institute* No. II, 1913) on Heating and Cooling Curves of Manganese Steel, the other by Messrs. Hadfield and Hopkinson (*ibid.*, No. I, 1914) treating of the Magnetic and Mechanical Properties of Manganese Steel. We then have brought together for this substance, the thermal, magnetic, and mechanical properties as well as the resulting microstructure, for the definite heat treatments described, and in particular after long annealing between 500° and 700° C.

Sir Robert Hadfield and Professor Hopkinson see in the properties found for manganese steel (of C = 1.25 and Mn = 12.5 per cent.) a demonstration of the untenableness of the allotropic theory of iron, whereas Professor Sauveur reaches the contrary conclusion, that some of the facts observed can be explained only in terms of the existence of beta iron in steel of this composition.

There is another possible hypothesis: namely, that these several series of experiments shed no light on the subject of the allotropy of iron, or at least as regards the existence of beta iron. However heretical it may sound, I am inclined to this last belief.

For steel with 1.25 per cent. carbon, we are in that portion of the Fe-C diagram, in any of its forms usually given, in which, for equilibrium conditions, or very slow heating and cooling, there is no beta iron possible, unless the improbable assumption is made that the carbon has combined mainly with the manganese rather than with the iron. But it is pre-

[3] Sir Robert Hadfield mentions 200 as the average ball hardness of quenched manganese steel.

[4] In submitting quenched manganese steel to similar heat treatments, *i.e.*, prolonged heating between 500° and 600° C. followed by slow cooling, Sir Robert Hadfield increased its hardness to as much as 495.

cisely when the equilibrium conditions are approached in heating and cooling that those properties of manganese steel are found on which are based the arguments of Messrs. Hadfield and Hopkinson on the one hand and of Professor Sauveur on the other. It would appear that if any beta iron were to be formed in heat treating this steel, it would be expected to be found only after relatively rapid cooling due to the non-homogeneity of the mass; but my understanding of the claims of Professor Sauveur is to the effect that beta iron is supposed to be liberated on long annealing (or tempering) between 500° and 700° C. after rapid cooling, but to be practically non-existent for rapid cooling and heating.

It would appear reasonable to attribute to the 12.5 per cent. of manganese here present—in its relation to the iron and carbon—the major rôle in the phenomena observed, although the effect of high carbon with little or no manganese in producing somewhat similar but less intense effects is not to be neglected, as pointed out by Messrs. Hadfield and Hopkinson.

The fact that the thermal and magnetic A_{123} point of this manganese steel is lowered from some 700° C.—evidently by the manganese—to liquid-air temperatures, or even suppressed, for ordinary rates of cooling, and is recovered, as shown by Messrs. Hadfield and Hopkinson, by long heating between 500° and 700° to nearly the normal value for carbon steel, or to 674° C., would seem to indicate that it is the manganese which is mainly responsible, this element acting in its well-known deterrent capacity and perhaps otherwise also.

It would be of interest to take (which does not seem to have been done) the cooling curve of this steel immediately after long annealing at 500° to 700° C.; the Ar_{123} point should then be found as sharply marked somewhere between 670° and ordinary temperatures as is the Ac_{123} point at 674° C., found by Messrs. Hadfield and Hopkinson. That the Ar_{123} transformation must have taken place is evident, since the steel is magnetic when cold after annealing at 500° to 700°; and the transformation from the non-magnetic to the magnetic state certainly involves the same quantity of heat and is probably as sharply located as the reverse phenomenon, although perhaps not at the same temperature, as is found for example in some nickel-chrome steels, for which Ar_{123} may be some 400° C. below Ac_{123}.

No argument is here offered for or against the allotropic theory of iron, but it is merely desired to point out why the existence or non-existence of beta iron would appear to play no rôle in defining the properties of this manganese steel, when equilibrium conditions of heating and cooling are approached.

Dr. J. E. Stead, Middlesbrough, England (communication to the Secretary*).—In discussing the allotropic theory I think that every

* Received Dec. 7, 1914.

metallurgist of repute agrees that there are two allotropic modifications of iron, the alpha and the gamma. The question always in doubt, and never fully answered by Osmond, is whether there is a third or a beta modification. The strong magnetic properties of hardened carbon steels always have been a stumbling block in the way of the beta advocates.

I have regarded with the greatest respect the experimental work of Osmond and his reasoning, which always were of the highest order, but after all that has been said for the beta theory one could only accept it tentatively as a reasonable hypothesis.

The peculiar properties of manganese steel depend on the carbon. Take carbon away, and, no matter what the heat treatment may be, it always remains magnetic. Manganese may be reduced to 2 per cent., but if the carbon is raised to 1.8 per cent., one obtains material which has some of the remarkable properties of manganese steel containing 12 per cent. manganese and 1.2 per cent. carbon.

I agree with the author that the magnetic properties in steel are coincident with the presence of alpha iron.

Austenite in cold steel has no martensite structure, but this structure, as the author has said, can be produced by annealing austenite. The coarseness of that structure depends on the temperature and time of heating before quenching. It is fine if the temperature has been high and of short duration; coarse if the preheating has been high and prolonged.

The structure developed by annealing austenite at about 600° C. is evidence of the crystalline structure of the austenite, which, being homogeneous, is not revealed by the ordinary etching reagents. Annealed austenite and its equivalent in quenched pure carbon steel have the martensite structure readily revealed by etching, one part along the cleavages being more readily attacked than other juxtaposed portions—a proof to my mind of non-homogeneity, a difference from austenite in possibly both chemical and physical constitution.

Very coarse martensite can readily be split along its cleavage planes, and in my opinion these same cleavage planes pre-existed in the metal when in the pure austenite state, but in that state could not be readily separated owing to the ductility of the whole mass. If we accept this basis that martensite is not homogeneous, what then is the partial change that occurs in austenite which is coincident with extreme hardness? Only one thing is certain, that one part of the martensite is magnetic, and therefore contains alpha iron; it may also contain, and probably does contain, some austenite.

The author, while concluding that the hardness of quenched and reheated manganese steel is due to the formation of a considerable proportion of beta iron, does not, I think, bring any proof excepting that of extreme hardness to show that any beta iron is really present. Is it not possible that, if there is any truth at all in the intermolecular pressure

theory, the expansion which occurs when a portion of the gamma iron changes to alpha, may be responsible for the hardness? It would, of course, be impossible by the microscope or any available means to discover the alpha iron, or the intermolecular pressure, but it seems quite as reasonable, if not more so, to assume that the hardness is due to interstress between alpha and gamma molecules rather than to beta iron + gamma iron + alpha iron.

I quite agree with the author that all hypotheses should be regarded with respect, and none of them should be dismissed without most careful consideration. I would also point out that nobody at present has sufficient data to justify any dogmatic conclusion being formed. What we want is more data, more research and more experimental evidence.

Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron

BY ALBERT SAUVEUR, CAMBRIDGE, MASS.

(New York Meeting, February, 1914)

In an important paper on The Critical Ranges of Pure Iron[1] presented at the May, 1913, meeting of the Iron and Steel Institute, Professor Carpenter reports and illustrates the results obtained by him in determining the critical points of some very pure electrolytic iron sheet about 0.01 in. thick. The iron strip was rolled into the form of a cylinder $1\frac{1}{8}$ in. long and $\frac{11}{16}$ in. wide, weighing 42 g., the thermo-couple being inserted in a small hole in the center. The points Ac3 and Ar3 were readily detected, as well as the point Ar2, but the point Ac2 was absent from every heating curve. The author concluded from this occurrence that the conception of A2 as an independent allotropic change must be abandoned.

At the October, 1913, meeting of the American Institute of Mining Engineers, Dr. G. K. Burgess and J. J. Crowe read a paper on The Critical Ranges A2 and A3 of Pure Iron[2] in which they described some extremely important experiments conducted by them at the Bureau of Standards, Washington, D. C. They tested Professor Carpenter's electrolytic iron and detected a sharp and well-defined absorption of heat corresponding to the point Ac2. Professor Carpenter's failure to detect this point is explained by them on the ground that unless the iron be in a single piece entirely surrounding and in contact with the thermo-couple junction, the thermo-couple will integrate the irregular progress of the heat through the sample and the curves will lose their sharpness. In discussing Messrs. Burgess and Crowe's paper, Professor Carpenter accepted this explanation as probably accounting for the absence of Ac2 in his heating curves.

Professor Carpenter had the kindness to supply me with some of his electrolytic iron. Heating and cooling curves of this metal were taken in the metallographic laboratory of Harvard University by F. C. Langenberg by means of an electrically heated tube furnace and a Le Chatelier-Saladin self-recording thermo-electric pyrometer, as constructed by Pellin, of Paris. The electrolytic sheet was hammered tightly around the

[1] *Journal of the Iron and Steel Institute*, vol. lxxxviii (1913, I), pp. 315 to 326.

[2] *Trans.*, xlvii, 665 (1913).

thermo-couple. These curves are shown in Figs. 1 and 2. The sharp occurrence of the point Ac2 at 765° C. should be noted.

In view of the recent and extraordinary attempt of Professor Benedicks[3] to account for the presence of the Ac2 point in Dr. Burgess's samples of electrolytic iron on the ground that the latter used an alternating current for heating his furnace, it should be noted that a *direct* current was used in obtaining the curves reproduced here.

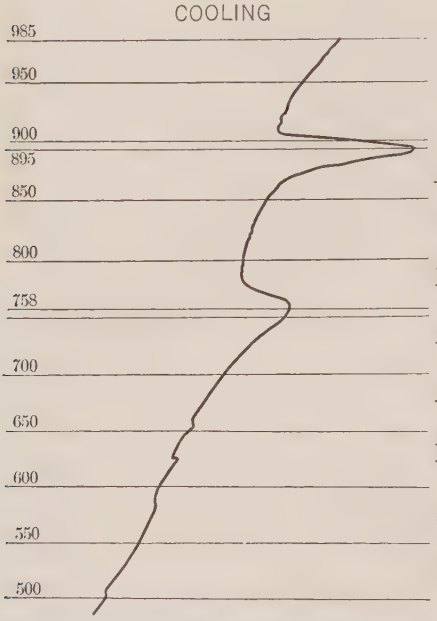

Fig. 1.—Cooling Curve of Professor Carpenter's Electrolytic Iron.

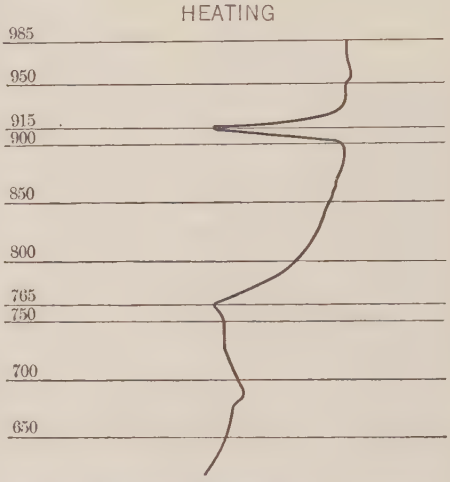

Fig. 2.—Heating Curve of Professor Carpenter's Electrolytic Iron.

Scientific speculation is not only permissible, but highly commendable and fruitful, provided it rests on a substantial foundation of experimental facts. Scientific speculations, on the contrary, however brilliant as intellectual gymnastics, if they lack such support are merely destructive, and when indulged in by investigators of repute, their ill-founded views are likely to be accepted by others but partly informed or biased, and the progress of science be retarded.

The existence of A2 in pure iron as an independent point is no longer a debatable question. Whether this point is or is not an allotropic point appears to depend upon our conception of allotropy. If it be insisted that an allotropic point necessarily implies a crystallographic change, or, in other words, that polymorphy and allotropy are synonymous, then it may be claimed, with reason, that A2 is not an allotropic point, since crystallo-

[3] *Trans.*, xlvii, 727 (1913).

graphic differences between beta and alpha iron have never been discovered. On the contrary, if, more rationally in my opinion, we believe that an abrupt heat evolution on cooling and a corresponding heat absorption on heating, when occurring in a pure element without change of state, must be taken as an evidence of allotropy, then A2 must be an allotropic point. Are not such thermal disturbances due to spontaneous changes of internal energy necessarily caused by molecular changes; *i.e.*, in the absence of change of state, by allotropic changes? We are strengthened in our belief in the allotropic character of the A2 point by the discontinuity in many of the physical properties of iron occurring at that point. It has been contended by those who doubt the allotropy of A2 that the absence of any expansion at the Ar2 point and of contraction at Ac2 argues strongly against allotropic change. Messrs. Burgess and Crowe, in the paper already mentioned, discuss this point as follows:

"Both from the work of Charpy and Grenet and of Broniewski, observations on the expansion of iron give no indication of the existence of A2. The former place the contraction observed as lying in the interval from 860° to 890° for iron with 0.03 carbon, while the latter finds it above 950° for electrolytic iron.

"This disagreement here is great enough to again raise the question, as was done by Le Chatelier in 1899, of the speed and other conditions of heating, and also whether the sensibility of the methods used was great enough to detect A2. Even its non-existence, in so far as expansion is concerned, would prove only that iron immediately above and below A2 has the same coefficient of expansion, or simply that the A2 transformation is unaccompanied by appreciable change in volume, a not unreasonable possibility. The more recent experiments of Rosenhain and Humfrey would appear to indicate a slight volume change accompanying A2." It should also be recalled that Osmond took exception to Messrs. Charpy and Grenet's conclusions drawn from their own experiments as to the absence of dilatation at Ar2. Osmond claimed that the curves obtained by Charpy and Grenet do indicate a dilatation at Ar2, the authors failing to notice it because the transformations not being sudden, the expansion likewise is gradual, whereas they were looking for sudden dilatations only.

Discussion

H. C. H. Carpenter, London, England.—I cannot pretend to be entirely uninterested in this paper but as beta iron has had plenty of discussion lately, I will be as brief as possible. The position really is this—that Professor Benedicks's theory that beta iron could be explained as a solution of gamma iron in alpha iron has given rise to a number of researches, and although it appears to me now that the theory, at any rate in the form in which it was put forth, is untenable,

yet even so, I think that Professor Benedicks was quite justified in his original paper, because undoubtedly the researches have advanced to this stage.

It is very curious that there never has been any serious disagreement regarding the meaning of the A1 and A3 points. And although in this case widely different apparatus was used, and apparatus of different sensitivity, there has been on the whole very good agreement with regard to the results; but we cannot even to-day say that that is the case with regard to A2.

I desire to refer to a statement in Professor Sauveur's paper, which reads as follows:

"On the contrary, if, more rationally in my opinion, we believe that an abrupt heat evolution on cooling and a corresponding heat absorption on heating, when occurring in a pure element without change of state, must be taken as an evidence of allotropy, then A2 must be an allotropic point."

If we test the break on the curve at A2 by that rule, then we are in this difficulty—that Professor Burgess's curves which were presented at the last meeting show, as I pointed out in the discussion, a sharp break at 768° on heating—that is, at exactly the same temperature as on cooling—but the direction of the curve indicated that there was either an evolution of heat at that temperature, or else the cessation of the absorption of heat, both of which would have the same result on the form of curve. Undoubtedly, on cooling at Ar2 we also have an evolution of heat at 768°. You cannot show, in my opinion, that those things are, strictly speaking, reversible; for the whole criteria of allotropic change is that you have an evolution of heat on cooling, and if that is so, it must be an absorption of heat on heating, but, as I have just pointed out, it is not an absorption of heat on heating, at any one point, but an evolution or a loss or cessation of the absorption of heat at that point. But, as was agreed upon by various protagonists in the discussion at that meeting, there is undoubtedly an absorption of heat going on over a lower range of temperature, and in my opinion a study of the matter shows that fact.

If we look at Fig. 2, we see that the curve breaks away at about 680°, slopes to the right, and there is that movement at the left, which, I take it, corresponds to the absorption of heat, which extends from 680° to 765°, or, roughly speaking, from 80° to 90°. My own results show that absorption in the same way. Indeed, the contention in London at the May meeting was that my own curve showed what they said it did not—that is, an absorption of heat.

My position is really this, I am convinced, and I owe it to this meeting to say so, and I owe it to the subject as well—I am convinced that Professor Burgess has made good the discontinuity in the curve and has borne out Professor Sperry's results at about 768°. The interpretation of the

curve is not yet clear. It does not appear to be exactly the reverse of the curve on cooling, because it is not an absorption of heat. We have in that connection to remember the recent work of the Japanese writer Honda, in which his conclusion, quite independent of these papers, is that the absorption of heat in this range of temperature is connected with magnetic change. However that may be, I am convinced they place the point of discontinuity of the curve at about 768°, and future research must show us what is the interpretation to be given to that result.

Henry M. Howe, New York, N. Y.—This beta iron Osmond used to call *Decipium*, just when you thought you had found it, it was not there. The case is somewhat comparable to that of the planets Neptune or Saturn, if either of them were not visible. Suppose Neptuné were invisible and could not be seen, to prove its presence would be extremely difficult, but its presence would be susceptible of proof from the perturbation of the other planets. In the same way, this beta iron has not been isolated—it may not have been isolated, and it may not be possible to isolate it, but we accept the evidence of its presence from the perturbations which it causes in all sorts of phenomena. The two strong perturbations difficult to understand on any theory other than the existence of beta iron, are the excessive hardness, and the transition form between gamma iron and alpha iron. No matter how you arrive at that transition form, you have arrived at that transition form by cooling rapidly, so as to catch the metal in transit between gamma and alpha, and you get extreme hardness. That is the way we get our hard cutting tools, and we get the extreme hardness of hardened steel. If, on the other hand, you bring about that intermediate state of an intermediate degree of carburization—if you take soft nickel steel and carburize it part way toward the gamma state, you get it intensely hard, and if you carburize it further and get it over into the gamma state, it becomes soft again; that is to say, whatever way you bring it about, if you catch the material between gamma and alpha, it is much harder than either gamma or alpha, and it is so much harder that it is difficult to explain it. It is not impossible, but it is difficult to explain it, and the explanation seems to be that that extra hardness is simply due to the solution of gamma in alpha.

It is difficult to understand why the solution of gamma in alpha should be so much harder than either gamma or alpha—gamma iron itself is a solution of cementite in gamma iron as we know it—when we dissolve further alpha iron in that solution, according to Professor Benedicks—and the reason why these intermediate states are harder than gamma iron is that alpha iron is dissolved in what was a solution; that is to say, starting in with a solution of cementite in austentite, and dissolving into that solution alpha iron, we must, to explain that theory, dissolve alpha

iron in that solution and we must increase the hardness of the solution—I think it is easier to believe and more natural to believe, and after all we take theories which are easier and more natural—more harmonious with the general aspect of nature—it is easier to believe that that extra hardness is due to the presence of this perturbed condition in beta iron, than to suppose it is simply due to the solution of alpha iron into a softer material, into a solution which already existed.

George K. Burgess, Washington, D. C. (communication to the Secretary*).—I am glad to see that Professor Sauveur has thought it worth while to examine with a recording apparatus the heating and cooling curves of pure iron and to note that, using a furnace supplied with *direct* current, he substantially confirms the results obtained by Mr. Crowe and myself using *alternating* current heating and also *gas* heating.[4]

We have since had the pleasure of demonstrating to Professor Carpenter, on the occasion of his recent visit to the Bureau of Standards, with a 0.4-g. sample of his electrolytic iron, the independent existence and location of A2 and A3 both on heating and cooling.

It appears, therefore, to be abundantly proved to the satisfaction of all who have recently partaken in the experimental work on the subject, that, as Professor Sauveur states, "The existence of A2 in pure iron as an independent point is no longer a debatable question."

Professor Henry Le Chatelier, Paris, France (communication to the Secretary†).—I beg to offer the following observations on the paper of Professor Sauveur:

1. The suggestion of Professor Benedicks, that the electric heating of furnaces may play a certain part in the development of the point A2, deserves experimental test. It certainly does not permit an absolute denial of the existence of the thermal phenomenon, for this is equally proved in cooling, during which there is no electric current at all. It is, however, perfectly logical to admit that the disappearance of energy due to the loss of magnetism in an electric field may liberate a certain quantity of heat. It would be necessary to compare the cooling of the same sample inclosed in a tube cooled on the outside by circulating water, and surrounded by a solenoid permitting or preventing, at the will of the operator, the passage of the current, which would then produce no thermal effect upon the sample, by reason of the intervening envelope of water.

2. I feel it a duty to repeat here, what I have often said already, that the point A2 cannot be regarded as marking anything like a change of allotropic state. The change at A2, which exists in all ferro-magnetic

* Received Mar. 23, 1914. † Received Mar. 26, 1914.

[4] *Trans.*, xlvii, 665 (1913).

bodies, differs from allotropic changes in being continuous, exactly like the progressive change of the properties of molten sulphur. It is demonstrated in a certain way for nickel, by the measure of dilatation. Moreover, the change is almost rigorously reversible; there is no notable retardation in the transformation, ascending or descending, while the transformation A3 and all the allotropic transformations suffer retardations which are sometimes enormous.

Sir Robert Hadfield, London, England (communication to the Secretary*).—I have read with much interest Professor Sauveur's paper, Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, and while regretting not being able to support the conclusions presented from the allotropist's point of view, yet I trust the following statements may be of service in helping to elucidate the problems referred to. The reasonable and moderate expressions of opinion so ably presented by Professor Sauveur show that he is willing to try to look fairly at both sides of this important subject.

Although the paper is only a brief one, yet the curves shown in it are of considerable interest, and raise what may be termed momentous questions.

Although mention is not made of the exact composition of the electrolytic iron experimented upon—for example, was it free from hydrogen, as no doubt it was, and assuming that it was—we have then to consider that the material in question, representing pure iron, shows a change point at that part of the heating or cooling curve known as the A2 point, which Professor Sauveur considers, as I understand, represents an allotropic change in the iron.

On the other hand, Dr. G. K. Burgess and J. J. Crowe, in the last paragraph of their reply to the discussion on their paper, The Critical Ranges A2 and A3 of Pure Iron, read before this Institute in October, 1913, state that if the A2 point could be proved to be due entirely to the magnetic change point, then it would appear that beta iron could not exist.

If the recent work by Prof. K. Honda on Transformations of Iron and Steel at High Temperatures, which appeared in the *Science Reports of the Tokyo Imperial University*, is correctly understood, this very demonstration seems to have been achieved. Professor Honda clearly states in the research just mentioned that the A2 point is due to disengagement of heat associated with the magnetic change. His words are as follows:

"With a refined method of measurement it was found that in iron, nickel and cobalt at the magnetic transformation, a distinct heat evolution took place. The heat evolution extended over a wide interval of temperature. With iron,

* Received Mar. 27, 1914.

steel, and nickel, the transformation is accomplished with a low intensity of field in a small interval of temperature, the interval increasing with rising field intensity. The author [Professor Honda] therefore arrives at the conclusion that in pure iron only two states, alpha and gamma, occur. The magnetic transformation hitherto termed alpha to beta is not a true phase-transformation, but a change of property as the result of temperature. Nickel and cobalt have only a single state in the solid."

Does not this, therefore, rather answer Professor Sauveur's very fair-minded comments, when he says:

"If it be insisted that an allotropic point necessarily implies a crystallographic change, or, in other words, that polymorphy and allotropy are synonymous, then it may be claimed, with reason, that A2 is not an allotropic point, since crystallographic differences between beta and alpha iron have never been discovered"?

It is true he adds that:

"On the contrary, if, more rationally in my [Professor Sauveur's] opinion, we believe that an abrupt heat evolution on cooling and a corresponding heat absorption on heating, when occurring in a pure element without change of state, must be taken as an evidence of allotropy, then A2 must be an allotropic point. Are not such thermal disturbances due to spontaneous changes of internal energy necessarily caused by molecular changes; *i.e.*, in the absence of change of state, by allotropic changes?"

It seems to me that Professor Honda has answered Professor Sauveur's question. Therefore, it appears to me most positively proved that the A2 change is not a definite evidence of an allotropic point.

It must not be forgotten that those who have been unable to accept the allotropic theory have had to question and contest the following statement:

"I conclude that hardened steel owes its properties principally to the presence of beta iron, which is hard and brittle by itself at the ordinary temperature."

The above is not a statement by one of no importance. It is a clear and definitely presented statement by one of our greatest masters of metallurgy, the late M. Osmond, whom we all respected. While allotropists have made certain changes in the theoretical presentation of their case, yet, as it is generally understood, those representing this line of thought still claim the above.

Probably some of this trouble has arisen for the reasons pointed out by Professors Edwards and Carpenter in their interesting paper on The Hardening of Metals, being read before the Iron and Steel Institute at the May meeting this year, namely, that "At a later period it was considered by Osmond, Roberts-Austen, and others who took up the views of the so-called allotropists, that the Ar2 point was quite distinct from the Ar3 point, and they came to regard the Ar2 point as corresponding to an allotropic inversion of the iron." Then, as these two gentlemen point out, "From that time onward, the beta theory as it is applied to the hardening of steels really became an entirely different theory, but unfortunately the same nomenclature was retained."

If allotropists would only define more clearly what is really now meant, it would be very helpful. There are, however, many different points of view among the allotropists. I do not mean to say for one moment that the carbonists can offer a clear solution of the problem, but the latter seem to me to come more nearly to the facts and phenomena noticed in the hardening of steel.

Incidentally, the following may be of interest: It may not now be remembered that it was probably owing to certain views expressed in my first paper on Manganese Steel, read in 1888,[5] that an allotropic theory exists to-day. In my original paper referred to, I made the following statement, under the section headed "Influence of Carbon upon Manganese and Iron":

"If manganese were the sole cause of the hardness noticed, it would naturally be supposed that the latter would be much harder; this however is not so, and the apparent anomaly occurs that when the percentage of iron (Fe), known to be a soft metal, diminishes, and is replaced by the supposed harder metal manganese (Mn), the hardness of the alloy decreases. This seems to prove that manganese must not be considered as the sole influence or cause of hardness. Even in those specimens containing as high as 20 per cent. Mn, and in which further modifications may occur to complicate the question owing to the carbon reaching higher amounts, 1.5 to 2 per cent., the hardness is less than that of the 4 per cent. Mn steel in its cast state.

"That iron (Fe) itself is a soft metal is well known, but it is worthy of consideration, at any rate, whether it possesses the power of becoming intensely hard, either by some peculiar form of crystallization, as discussed further on, or owing to its forming certain combinations or alloys. In other words, whereas the hardness of steel has hitherto been entirely attributed to the bodies present in much smaller proportions as compared with the iron, may it not be possible that the part played by iron (Fe) is equally, if not more, important, perhaps owing to some change of form of the iron itself not hitherto suspected, as in the case of the intensely hard 4 per cent. manganese-steel, where carbon cannot possibly be the cause, and where manganese would not appear to be the sole cause? . . . It consequently becomes more important to ascertain whether the manganese itself causes the hardness in this steel, or is only a means whereby iron is caused to take this nature."

In the paper entitled Iron Alloys with Special Reference to Manganese Steel,[6] it will be noticed that I had already taken up a new modified line of thought as compared with that suggested in 1887–88.

Now why was it I abandoned the theories advanced, it should be remembered, so far back as 1888? My reply is that in the first place I found that some of the beliefs then expressed were founded on data which at the time appeared to be correct. I afterward found, however, that as regards the determination of the carbon percentages actually present in one or two of the specimens upon which I based certain conclusions, the analyst had underestimated the percentage present.

[5] *Proceedings of the Institution of Civil Engineers*, vol. xciii, pp. 72, 73 (1887–88).
[6] *Trans.*, xxiii, 148 (1893).

That is to say, the carbon percentages were, by more correct modern analytical methods used for determining carbon in alloy steels, found to be considerably higher, that is relatively to the smaller amounts present, 0.4 to 0.5 per cent., instead of about 0.3 to 0.4 per cent., as then estimated. The increasing and rapid influence of carbon over 0.35 per cent. is well known. Consequently, with corrected figures before me, some of the original remarks did not hold good, and the disregard of the influence of certain somewhat higher percentages of carbon had been too much.

Singular to say, Sir W. C. Roberts-Austen, in contributing to the discussion on my paper in 1888, said "he thought that the author had a little under-rated the influence of the small amount of carbon which the manganese iron contained." In other words, this scientist, although afterward a strong allotropist, mentioned the very point I and others have so strongly emphasized: namely, that we have arrived at the conclusion that it is rather to the influence of the carbon present than to any allotropic change in the form of the iron present, that we must look for the correct explanation of the many peculiarities met with in steel and its alloys. As time passes, I am still more convinced of the correctness of our attitude.

It should not be overlooked that your distinguished metallurgists, Mr. Metcalf and Professor Langley, in 1893 read a paper before the American Society of Civil Engineers, vol. xxvii, p. 382, in which they gave a brief statement of the carbon theory of hardening, to which paper the reader is referred for fuller information. Briefly, their statement was that carbon alters the molecular aggregation, and the resulting strained grouping is brought about within certain temperature limits; also molecular stresses are set up, which produce hardness in a similar manner to that of cold working.

This view has been further elaborated in a most important paper by A. McCance, B. Sc., A. R. S. M., entitled A Contribution to the Theory of Hardening, to be read before the Iron and Steel Institute at the May, 1914, meeting. The writer considers this one of the most important contributions to the metallurgical world on this subject. It is therefore doubly of interest to find, to use the words of the late respected Mr. Metcalf and Professor Langley, that hardening is the result of molecular stresses, which produce hardness is a similar manner to that of cold working. In this paper by Mr. McCance it is stated:

" Based on the experimental work of Curie, the electronic theory of magnetism was first put on a sound basis by Langevin, and has since owed most of its development to Weiss. Every atom is supposed to have a definite field of its own, caused by its internal system of revolving electrons, and under the action of an external field the molecules (or atomic aggregates) tend to set themselves all in one direction, giving rise to a definite intensity of magnetism which is the total effect of all the molecular magnets in the substance added together. In gases and liquids they are prevented

from all turning in the one direction by forces and collisions depending on the translational velocity, and in solids other forces which may be called 'crystalline' forces come into play. Instead of preventing the molecules from turning round under the action of an external field, these crystalline forces may actually assist them to turn, and it has been conjectured that this is the special action which goes on in the so-called 'ferro-magnetic' bodies, which are bodies like iron that become strongly magnetic under weak external fields. As on heating a liquid, when a temperature is reached at which the forces between the molecules become insufficient, the liquid becomes a vapor, the forces causing ferro-magnetism gradually become inoperative on heating until a temperature is reached at which they disappear rapidly and the body becomes non-magnetic. . . .

"In iron, then, the property of loss of magnetism cannot be used as a proof of allotropy—it is no sudden transition, but an effect depending only on temperature. .

"Accompanying every alteration in the magnetic intensity, there is a variation in the specific heat, . . . proportional to the rate of change of the square of magnetic intensity at any temperature, and it has a maximum value always at a temperature slightly below the magnetic transition temperature. . . . It begins gradually and rises to a maximum at about 760° C. for pure iron, becoming zero as the iron becomes non-magnetic; and the important point is that this change in the specific heat cannot be distinguished in a cooling curve from a true absorption of heat due to an allotropic change. In the author's opinion this confusion has actually taken place at the Ac2 point in iron. The variation is quite large enough to account for all the thermal change, and the gradual character of the Ac2 point is entirely in accordance with this nature.

". . . The temperature at which the non-magnetic condition commences has no connection with the true allotropic ranges, and takes place quite independently of them—thus pure iron becomes paramagnetic at 820° C. (Curie), which is 100° below the Ac3.

"There is this great difference between a pseudo-change point caused by a sudden change in the specific heat and a true liberation of heat caused by allotropy. When the temperature is rising very slowly, the effect due to specific heat is masked in a differential heating curve, since the difference measured by the differential galvanometer is proportional to the product of the rate of heating and the specific heat. With a true thermal change point, however, the amount of heat liberated is independent of the rate of heating.

"The sharpness of the A2 point is much less on heating than on cooling, . . . as the change is more gradual when approached from the low temperature side than from the high temperature side.

"The failure of Carpenter to notice the Ac2 point in pure iron, must be ascribed to the nature of the change being as above described. . . .

"The problem of beta iron is thus reduced to a very simple one, and without making any assumptions regarding the nature of alpha or gamma iron, while taken with the evidence produced by other experimenters, the conclusion is undeniable that beta iron as a separate allotropic modification has no existence, since it is merely alpha iron which has lost the properties of ferro-magnetism."

Professor Weiss believes that by the electrical disposition of what he theoretically terms "magnetons" he is able to offer more satisfactory explanations of magnetic phenomena than by any other theories. He also appears to consider that the increase of magnetons, with their important bearing upon magnetic properties, is independent of allotropic and perhaps even chemical modifications, of the metal under examination.

I have dealt with this more fully in the joint paper by Professor Hopkinson, F. R. S., and myself, to this Institute.[7]

Carl Benedicks, Stockholm, Sweden (communication to the Secretary*).—I quite agree with the author in laying stress on "a substantial foundation of experimental facts." However, I have the impression that this foundation is not very much consolidated by repeating, apparently with less accuracy, or at least with no greater accuracy, than by previous workers, exactly the same style of experiments—however interesting in themselves. From Professor Carpenter's and from Dr. G. K. Burgess and J. J. Crowe's important work, we know that occluded gases have the effect of rendering the thermal changes in the A2 region much more considerable than they are when the gases are removed. Professor Sauveur's curves show an evolution of heat, or absorption, at A2, which in comparison with that at A3 is much more considerable than in Professor Carpenter's or in Messrs. Burgess and Crowe's curves, and we find no indications how, or even if, the gas content, so important for the thermal irregularity at A2, was lowered.

As for increasing the experimental foundation of our allotropy views, the writer considers it more appropriate to repeat earlier experiments with an accuracy much higher than that formerly used. This is the reason why the writer, likewise on iron kindly supplied by Professor Carpenter, lately has repeated the dilatation work of Svedelius, Le Chatelier, Charpy and Grenet (and Broniewski), using a compensation method which, owing to special arrangements, gave a sensibility about 1,000 times greater than that used at the work of the last-named authors. Simultaneously, on the same material, magnetometric and magnetostrictional measurements were carried out. The result, which might interest Professor Sauveur, is that there is *not the slightest trace* of discontinuity at A2. If there is a discontinuity, it is quite surely less than three-thousandths of the discontinuity at A3. Even the continuous length decrease which was found to occur together with the disparition of the ferro-magnetism, and which is a necessary effect of magnetostriction, is a rather weak one.

Until some still more fundamental work is published than J. Willard Gibbs's papers on the Equilibrium of Heterogeneous Substances, we must consider this phase rule as a reliable standard. In order to prove that an *allotropic point* occurs at a definite temperature, according to Gibbs's theory, it is a necessary condition to prove that *two phases occur* at that temperature. This is done by proving that a real discontinuity occurs in the properties of the substance at a definite temperature, *at least in a single case*. It is not in harmony with Gibbs, and too narrow, to select

* Received Apr. 2, 1914.

[7] This volume p. 476.

as a criterion the property of crystallization, as was proposed by Messrs. Burgess and Crowe in their important paper. Any property is to be accepted as proof of allotropy point, provided it gives a clear discontinuity; that is to say, two different values of the property at one and the same temperature. Now, no such discontinuity, in the strict sense of the word, has ever been proved to occur at A2, and since the dilatation experiments, denoted by Messrs. Burgess and Crowe as so important as to have a crucial bearing, have been found not to give the slightest discontinuity, we may be very sure in concluding that *A2 is not an allotropic point.*

I have some slight hope that Professor Sauveur will judge of the writer's theory somewhat more favorably than as yet, after reading the paper which the writer has prepared for the Iron and Steel Institute's spring meeting. Therein are also to be found some reliable measurements on ferro-magnetic mixtures (magnetite, iron, nickel) giving an experimental support to the writer's attempt[8] to explain the fact that the disparition of ferro-magnetism occurs at a much lower temperature than the allotropic transformation point, A3.

N. Belaiew, St. Petersburg, Russia (communication to the Secretary*).—The few remarks I have to add to the most valuable and instructive paper of Professor A. Sauveur are in relation to the crystallization phenomena in iron-carbon alloys. Let us consider the so-called Widmanstättian structure with its characteristic sharp, straight lines; the first look at such a specimen leads us to the idea that a *sudden* precipitation of a proeutectoid element (ferrite or cementite) took place; but if the crystallization occurred suddenly the allotropic change must also be a sudden one and not a gradual one, covering a wide range of temperature, as Professor Benedicks argued.

On the other hand, the same crystallization phenomena show me only two distinct crystallographic bodies—the gamma iron and the alpha iron. The first (Feγ) is related to the structure of austenite—the "granula" or austenite grains; the second, with the pearlite grains. The network of the proeutectoid element shows us the first; the pearlite lamellæ, the second.

To express myself quite clearly, I say that only between gamma and alpha iron is there a sharp crystallographic difference to be detected; in other words, gamma and alpha iron are polymorphous bodies. But in my opinion different allotropic modifications do not necessarily imply a crystallographic change. From this point of view, not the crystallography of the studied modifications, but the change of internal energy, must be considered as the strongest proof of allotropy.

* Received Apr. 11, 1914.

[8] *Journal of the Iron and Steel Institute,* vol. lxxxvii (1913, I), p. 334.

As to this change, it is made clear principally by the heat evolution or absorption and we have to study these on our cooling curves.

The author's researches, as well as those of Dr. G. K. Burgess and J. J. Crowe, show us quite unmistakably that in pure iron A2 exists, always as *an independent point.*

This conclusion is very important and the reported curves are highly instructive, and I hope that everybody "before abandoning Osmond's theory, so satisfying because so simple and rational," will seriously and attentively study the paper of Professor Sauveur.

Albert Sauveur (communication to the Secretary*).—I am appreciative of the interesting and important discussion of my paper contributed by so many distinguished authorities. It is a great pleasure to find myself in substantial agreement with Professors Carpenter and Howe, with Captain Belaiew and Dr. Burgess. Professor Carpenter, however, is not yet convinced that the A2 point marks a reversible transformation, and, on that ground, he would question its allotropic character. I find it difficult to follow his contention that the thermal disturbances at A2 indicate an evolution of heat, both on heating and cooling. This seems quite inconceivable to me, for, thermodynamically speaking, such a phenomenon would be in the nature of perpetual motion.

Certainly Professor Howe's statement that it is easier and more natural to believe that the extreme hardness produced by preventing the complete transformation of iron-carbon alloys is due to the retention of a transition form of iron between the gamma and the alpha state rather than to ascribe it to a solution of alpha iron in gamma iron, or rather in austenite (itself a solution of the carbide Fe_3C in iron), cannot be disputed. We should resist unnecessary complications in scientific theories. As Professor Howe well puts it, the theories which should most appeal to us are those which are more "harmonious with the general aspect of nature."

Professor Benedicks contends that the cooling curves of Professor Carpenter's electrolytic iron, reproduced in my paper, add no information of value to the present controversy, because they were determined with less accuracy, or, at least, with no greater accuracy, than similar curves taken by Dr. Burgess. While I am quite willing to grant his claim, I had hoped that, in view of the fact that a direct current was used for heating the iron, my curves would have been considered as having some significance, owing to Professor Benedicks's claim that in Burgess's experiments the occurrence of the A2 point might have been caused by his having used an alternating current. I might also add that in my experiment the iron was repeatedly heated previous to the taking of the thermal curves for the purpose of expelling the occluded gases.

Professor Benedicks's statement that he was unable to detect the

* Received Apr. 13, 1914.

slightest trace of dilatation at Ar2 in Professor Carpenter's iron, even when using a method of extreme accuracy, is very important and will undoubtedly receive the attention it deserves. But does it prove the non-allotropic character of A2? While dilatation taking place abruptly during the cooling of an element may prove the existence of allotropy, does the absence of such dilatation prove its non-existence? An allotropic transformation does not, I believe, imply discontinuity in *all* of the properties of an element. If discontinuity is detected in some of its properties, is it not sufficient to prove its allotropy? Of course Professor Benedicks claims that "no such discontinuity, in the strict sense of the word, has ever been found to occur at A2," and, on that basis, he rejects the allotropic theory of the A2 point. Many investigators believe, however, that discontinuity in several of the properties of iron has been shown to occur at A2.

The absence of dilatation at Ar2, which Professor Benedicks uses as a weapon with which to deal a blow, which he considers final, to beta iron, might easily be turned against his own theory with apparently telling effect. The reasoning is as follows: (1) the point A3 marks an allotropic transformation; (2) the Ar_3 transformation is accompanied by a marked dilatation; (3) Professor Benedicks believes that the A2 point is part of the A3 transformation; (4) the Ar_2 point should therefore be accompanied by dilatation; (5) Professor Benedicks finds complete absence of dilatation at the Ar_2 point; (6) *ergo*, the A2 point cannot be part of the A3 transformation.

Professor Le Chatelier doubts the allotropic character of the A2 point for three reasons: (1) because a similar point is detected in all ferro-magnetic bodies; (2) because the A2 transformation is continuous; and (3) because the A2 transformation is "almost rigorously reversible." Taking these objections in order:

(1) The fact that the thermal point exists in all ferro-magnetic bodies, corresponding to the final disappearance of magnetism on heating, does not in my opinion argue against its allotropic character. It indicates a body capable of existing under two sharply distinct conditions, *i.e.*, a magnetic and a non-magnetic one, and implies the existence of a dividing line between the two. Is that not allotropic? Or will it be contended that the magnetic and non-magnetic conditions of an element represent one and the same allotropic condition? It is evident, as I have so often said, that the point in dispute must be settled one way or the other according to our definition of allotropy, and as to this, unfortunately, scientists entertain widely different views.

(2) Does the fact that the A2 transformation is continuous bar it from the allotropic class? To begin with, is it not somewhat misleading to describe it as a continuous transformation and *nothing more*, seeing that the end-point of that transformation is so intense and of such magni-

tude as to produce a distinct and spontaneous evolution of heat, indicative of a notable change of internal energy?

(3) As Professor Le Chatelier states, and as so conclusively shown by Burgess and Crowe, the A2 transformation appears to be rigorously reversible; that is, the Ar2 and Ac2 changes occur at exactly the same temperature. There is here a complete absence of hysteresis, whereas the A3 transformation and allotropic transformations in general, so far as they have been studied, always exhibit a temperature gap between the two opposite phases. In my opinion, however, this absence of hysteresis can hardly be considered as a proof of the non-allotropic nature of the A2 point. In elements, and in many other substances, the passage from the liquid to the solid state and the reverse change (solid to liquid) generally occur at exactly the same temperature, and may not these changes of state be considered as major cases of allotropy? Until we have a more intimate knowledge of the nature of the forces at work producing hysteresis, we can hardly claim that when those forces are absent or only feebly active there can be no allotropic transformation.

Replying to Sir Robert Hadfield's valuable criticism of my paper, I should say that the electrolytic iron used in my experiments was kindly supplied to me by Professor Carpenter and that it was similar to the iron used by himself in his investigation described in his paper on The Critical Ranges of Pure Iron with Special References to the A2 Inversions.[9]

The iron was repeatedly heated and cooled before taking the critical curves and, in my opinion, must have been practically free from hydrogen.

Referring to Sir Robert's quotation of a statement of Burgess and Crowe to the effect that "if the A2 point could be proved to be due entirely to the magnetic change, then it would appear that beta iron could not exist," it should be borne in mind that this conclusion of the authors is warranted only by their definition of allotropy, which they advanced in a tentative way as "an allotropic transformation is one accompanied by crystallographic change," a view which is by no means generally held by scientists, as will be apparent from Professor Howe's and Captain Belaiew's contributions to the present discussion.

Granting the accuracy of Professor Honda's work and conclusions, the allotropic or non-allotropic character of the A2 point still remains a question of definition. Professor Honda recognizes the existence of iron under three conditions, *i.e.* (1) gamma iron, non-magnetic and stable above the A3 point; (2) alpha iron, magnetic and stable below the A2 point; (3) non-magnetic alpha iron, stable between the A3 and the A2 points; and why should we cease to call this last variety beta iron? In which way has Professor Honda demonstrated the non-existence of beta iron?

Osmond's statement quoted by Sir Robert and reading as follows,

[9] *Journal of the Iron and Steel Institute*, vol. lxxxvii (1913, I), pp. 315 to 326.

"I conclude that hardened steel owes its properties principally to the presence of beta iron, which is hard and brittle at the ordinary temperature," still stands.

Sir Robert alludes to two extremely important papers recently presented before the Iron and Steel Institute in London, one of them by Professors Edwards and Carpenter, and the other by Mr. McCance. These contributions are the most important ones that have been offered for many years and I hope to discuss them at some length. It will suffice for the present to call attention to the fact that these two splendid arguments do not upset the allotropic theory of the hardening of steel. On the contrary, as I have said, they reinforce that theory. The basic claim of the allotropic theory is the existence of a condition between the gamma state and the alpha state and which when retained in the cold imparts great hardness to the steel. In the two papers referred to it is made evident that such a condition must exist and must be retained in order to explain the hardening of steel. It matters not, so far as the allotropic theory is concerned, what that condition really is; for convenience, it is called beta iron. The fact that it may be, according to Mr. McCance, alpha iron which has lost its ferro-magnetism, or interstrain alpha iron, or, according to Edwards and Carpenter, amorphous alpha iron, does not by any means eliminate the allotropic theory. It merely gives us more insight in regard to the possible nature of beta iron. It adds to our knowledge of beta iron and, therefore, rather strengthens its existence.

Captain Belaiew's very clear statement in regard to his conception of allotropy is very satisfactory indeed. It appears to me a more rational one than the view which considers a crystallographic change as a criterion by which to judge allotropy.

Notes on the Plastic Deformation of Steel During Overstrain*

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(New York Meeting, February, 1914)

§1. FOUR ASPECTS OF FLOW in the plastic deformation of steel by overstrain, such as punching, wire-drawing, tensile rupture, etc., are:

(1) The *inter-granular*, *i.e.*, the relative movements of the several grains, each as a whole;

(2) the *intra-granular*, *i.e.*, the relative movements of the masses of ferrite or cementite and pearlite within those grains;

(3) the *intra-pearlitic*, *i.e.*, the relative movements of the pearlitic ferrite and cementite; and

(4) the crystal unit slipping, *i.e.*, the travel of the minute units past each other, along the slipping planes, giving rise to the slip bands of Ewing and Rosenhain.

It may be objected justly that (2), (3), and (4) are in a sense "intra-granular," and that (4) is in a sense "intra-pearlitic." Hence clearness may be gained at times by calling these "Movements of the 2d, 3d, and 4th orders" respectively.

Tresca, and more lately Ewing, Heyn, Humfrey, Rosenhain,[1] and Stead have thrown great light on the mechanism of flow. We have sought chiefly to add to their discoveries by observing the intra-pearlitic

* A contribution from the Metallurgical Laboratories of Columbia University Received Jan. 20, 1914.

[1] Tresca: *Proceedings of the Institution of Mechanical Engineers* (London), 1876 and 1878. Cf. Unwin: *Testing of Materials of Construction*, 1910, p. 40.

Ewing: *Journal of the Institute of Metals*, 1912, No. 2, vol. 8, p. 4.

Ewing and Rosenhain: *Philosophical Transactions of the Royal Society* (London), A, 1900, p. 353.

Heyn, Martens-Heyn: *Materialenkunde*, II A, 1912, pp. 219, 259 *et al.*

Humfrey: *Philosophical Transactions of the Royal Society* (London), A, 1902, vol. 200, p. 225. Iron and Steel Institute, *Carnegie Scholarship Memoirs*, 1913, vol. 5, p. 86.

Rosenhain: *Journal of the Iron and Steel Institute*, 1904, No. 2, vol. 65, p. 335. *Idem*, 1906, No. 2, vol. 70, p. 189. *Engineering*, 1908, vol. 86, p. 340.

Rosenhain and Humfrey: *Journal of the Iron and Steel Institute*, 1913, No. 1, vol. 87, p. 219.

Rosenhain and Ewen: *Journal of the Institute of Metals*, 1912, No. 2, vol. 8, p. 149, and 1913, No. 2, vol. 10, p. 119.

See also Longmuir: *Journal of the Iron and Steel Institute*, 1913, No. 1, vol. 87, p. 219.

movement of the 3d order, and the movements of the 2d order in the very violent throes of punching.

We have examined microscopically longitudinal sections through the fractures of tensile test bars of steel of 0.21, 0.40, and 1.45 per cent. of carbon, Nos. 2, 3, and 7 of Table I, p. 546, the longitudinal section of a wire-rod of bridge-cable steel, which has been drawn part way through a die in the first draft of wire making, No. 5; and vertical sections through holes punched sometimes through, sometimes part way through, tank-plate steel, and steel of 0.40 and of 0.78 carbon, in some cases with the usual tight-fitting die, and in others with a die much wider than the punch itself, Nos. 4 and 6, or in short with both cylindrical and conical punching. We have examined also the deformation in the previously polished upper surface of the tank plate, caused by both cylindrical and conical punching, Figs. 9, 10, and 11. The tensile properties of the tensile test bars are given in Table I, and certain details of the punching are given in Table II, p. 546, including the linear and cubic shortage of the punched wad, etc.

§2. THE INTER-GRANULAR movements are readily seen on a previously polished surface soon after overstrain starts in. Galy-Aché[2] points out that, with the passing of the yield point, comes the development of grains like those developed by etching. The uplift of these grains[2A] on punching is seen clearly in Figs. 9, 10, and 11 of Plate 2. The displacements are greater in cylindrical, Fig. 10, than in conical punching, Fig. 9, which might well be foreseen; but there is a further and unexpected difference, that the matted area throughout which these inter-granular movements occur is wider in conical than in cylindrical punching. This probably represents the greater bending down of both upper and lower surface in conical punching. Further observations are needed to confirm this. Fig. 11 shows on a large scale one corner of such an uplifted grain.

§3. THE RELATIVE MOVEMENTS OF THE PEARLITE AND THE PRO-EUTECTOID FERRITE, in the punching of a $\frac{1}{4}$-in. plate of steel of 0.40 carbon with a $\frac{1}{4}$-in. punch, can be traced in Figs. 3, 4, 6, and 8 of Plate 1. Figs. 4, 6, and 8 represent the use of the customary tight-fitting die, or "cylindrical" punching, and Fig. 3 represents the use of a wider $\frac{3}{8}$-in. die, or "conical" punching.

In each case the punched hole is of two parts, the parallel sided and the diverging sided, or the polished "cylinder" and the matte "bell," Fig. 5, but there is this striking difference, that with the narrow die the divergence begins very near the exit end of the hole, whereas with the wide die it begins about one-third way down from the entrance end.

There is the further difference that, with the narrow die, Fig. 4, the

[2] Galy-Aché, *Rev. Métallurgie*, Memoires, 1913, vol. 10, p. 588.

[2A] We may surmise that the units which act as grains in this uplift represent what were the austenite grains before austenite-pearlite transformation took place. The massing of the ferrite would be greatest in the boundaries of these grains, and hence these boundaries would be the planes of least resistance.

elongation of the individual ferrite and pearlite masses into hairs or threads is more marked, but that the deformation has been confined to a narrower ring, than with the wide die, this latter difference reflecting the wider matting of the polished upper surface with the wide die. Thus with the narrow die the plate as a whole is less bent, but the pearlite and ferrite masses are more sharply bent, than with the wide die. Fig. 3 of course has no part comparable with the lower part of the edges of the hole in Fig. 4, in which the drawing out of the hairs has been extreme, for in Fig. 3 this part of the plate has been torn out in punching, and forms part of the wad.

Figs. 6 and 8, showing on a smaller scale the remaining region of cohesion and contact between wad and plate at stages during the progress of cylindrical punching, when taken in connection with Fig. 5, give some conception of the mechanism by means of which continuity is retained till the punch has moved nearly fully through the plate. This mode of flow seems like that of Tresca, a drawing out of the individual ferrite and pearlite masses into long hairs, but without implying great slipping of these masses as such, *i.e.*, without implying that the part of the pearlite hair with which a given part of a given ferrite hair is in contact at the end of the punching is a different part from that with which it was in contact initially. There has of course been such slipping in the movements of the fourth order, such exchange of partners, between the crystalline units of which the ferrite and pearlite masses respectively are composed, and no doubt a corresponding exchange on this same ultra-microscopic scale between the adjoining surfaces of ferrite and of pearlite.

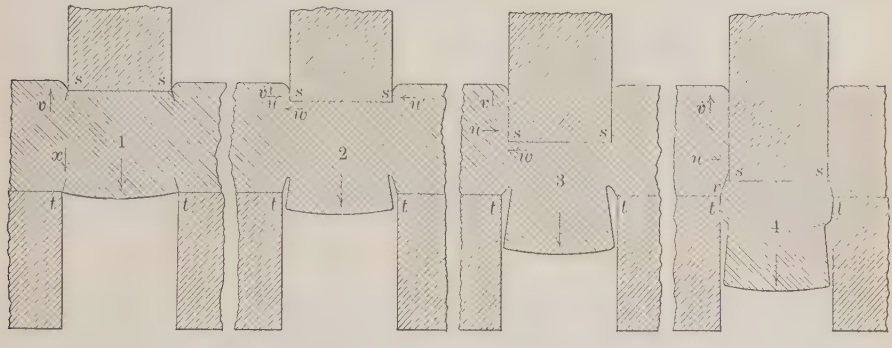
This is especially true of the surfaces of contact between ferrite and pearlite in those regions in which the degree of elongation decreases rapidly as we pass radially away from the imaginary prolongation of the cylindrical surface of the punch, yet perhaps not more true of those contacts themselves than of every neighboring imaginary plane parallel to them. Where the degree of elongation or travel thus varies radially to the punched cylinder, so that one infinitely thin walled cylinder lengthens more than that adjoining it, there is an infinitesimal slip of one past the other; and when for infinitely thin walled cylinders those of crystal-unit thickness are exchanged, there comes a like slipping measurable in crystal-unit thicknesses.

But though in this sense the actual slip of any one surface is probably very slight, yet the integration of the very many slips at the very great number of surfaces leads to a great total of slip and of slipping friction.

Moreover, this drawing out of a ferrite grain into a ferrite hair means an enormous increase in the ferrite surfaces, and hence an enormous degree of emergence of ferrite crystal units, which initially were in contact with ferrite on all six faces within a given ferrite grain, to the surface of that grain where one of the faces of each unit is in contact with pearlite.

In short, one face of each unit which has thus moved to the surface of a ferrite hair has, in that motion, exchanged contact with a ferrite particle for contact with pearlite, and yet has preserved continuity, hypothetically through the interposition of Beilby's fluid state.

§4. Tentative explanation of the movements in punching. Our inquiries were not directed specially to the flow which occurs during punch-

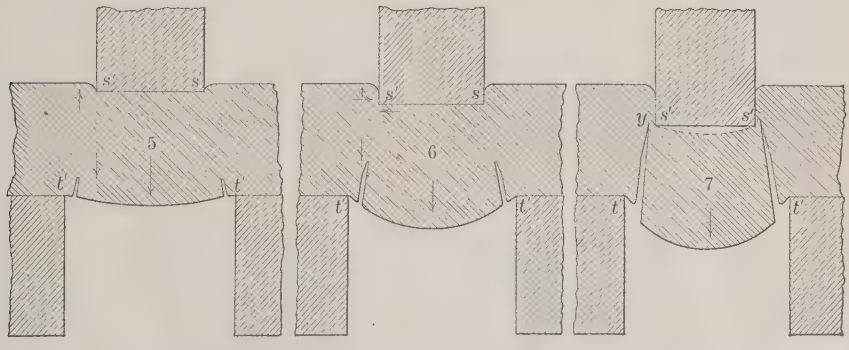


ss, the punch. *tt*, the die.

Fig. 1.—Probable Course of Flow in Cylindrical Punching.

ing, nor have we gone far enough to form firm convictions about them, but it seems well to record tentative inferences on this subject drawn from some rather striking phenomena.

The complete preservation of continuity, so beautifully shown by Tresca in the punching of ductile metals, is lacking in punching our



s', the punch. *t't'*, the die.

Fig. 2.—Probable Course of Flow in Conical Punching.

0.40 carbon steel. There has, indeed, been great drawing out of the ferrite and pearlite, especially with the narrow die, into the long hairs, alternately black and white, from *a* to *d* in Fig. 4. But the distortion has exceeded the ductility of the metal, and started cracks, which in some cases are only abortive, as at *b* in Fig. 4, where a bell like that at *a* has started, with the tearing out of a triangle now seen in black. From the

lower right-hand corner of this bell a crack runs out into the solid metal. Another abortive crack is seen about $\frac{7}{8}$ in. above the top of the wad in Fig. 8. In Fig. 5, *A*, two thin nearly vertical cracks are seen, following the slight taper of the wad itself, back into the solid metal. As seen on a larger scale in Fig. 6, one of these cracks looks as if it were about to be modified by the further movement of the wad. In Fig. 8 there is an obtuse-angled incision into the wad, about $\frac{5}{8}$ in. below the lower edge of the plate, on this scale. This incision may have been formed from such a crack as is shown in Fig. 6, by the continuing moving down of the wad, the acute angle which this crack turns toward the wad gradually opening out into the obtuse angle of Fig. 8.

A tentative explanation of the various facts is sketched in Figs. 1 and 2, representing stages in cylindrical and conical punching respectively. If the metal were a perfectly mobile liquid, then the pressure would be the same in every direction and at all points. And though, because of viscosity, the pressure decreases both as the distance increases from the corners of punch *ss* and of die *tt*, Fig. 1, and as the direction deviates from that of the punching, there is yet very appreciable pressure in all directions.

Thus, it is to the upward component, *v*, stage 1, of that pressure, that we may refer the forcing up of certain grains at the surface, shown in Figs. 9, 10, and 11, and the matting of its initial high polish. Again, it is to the centripetal component, *u* and *u'*, stages 2 and 3, of that pressure that we refer the high polish or schlickensides in the cylindrical part of the hole, both with cylindrical and with conical punching (Fig. 5, Plate 1). That polish indicates that, while the punch was moving down, the metal of the plate was forced strongly against it, as at *u* and *u'*.

In strong contrast with this polish is the matte surface of the bell, agreeing with our inference that the bell results from a crack starting at or near the corners *tt* of the die.

Cracks tend most strongly to start where the deformation is greatest, at the corners *ss* of the punch and at or near the corners *tt* of the die, and once started tend to be extended by the downward movement of the wad. But the conditions tend more strongly to this extending at *tt* than at *ss*, and in conical than in cylindrical punching, as we will now explain.

As regards starting a crack, the conditions at *t* and *s* seem to be much alike, in one case the pressing of a sharp edge into the metal and in the other the pressing of the metal against a sharp edge, with the difference that the concentration of pressure is greater at *s* than at *t*, as is readily seen if we imagine a plate 1 ft. thick punched with a punch only $\frac{1}{4}$ in. in diameter. This difference represents the centrifugal movement of part of the metal which lay initially in the path of the punch, *i.e.* its radial movement, resulting in bulging the plate somewhat in the neigh-

borhood of the hole, so that the volume of the wad is less than that of the hole.

As between conical and cylindrical punching there is the difference that the pressure on the die corners is weaker in the former at $t't'$ than in the latter at tt, both because in the former the pressure is spread out over a longer circumference, and because in the former the tendency to flexure is less at the die corner t', than at a point nearer the axis of the punch. This latter difference is readily seen if we imagine a $\frac{1}{4}$-in. punch driven into a $\frac{1}{4}$-in plate resting on a 3-in. die. Indeed the shoe-heel or burr, which projects down below the edge of the bell at the bottom of Fig. 2, suggests that here the crack started, not exactly at the edge of the die, but a little nearer the axis of the punch, either because of a local weakness, or because of the stronger bending moment there, or for both reasons. The direct measurements in Table II show that the final clearance between the die and the edge of the wad is about 0.05 in. At the time of the starting of the crack it was probably less, because the wad has probably narrowed as it bulged down in the further downward movement.

Apart from this, there is the difference beween top and bottom, that the propagation of the crack started at s is opposed by the strong radial pressure w and u in stage 2, and in the stages immediately following the starting of the crack, an opposition which has no full equivalent at t.

It is to this strong centripetal pressure that we refer the arresting of the cracks as at b in Fig. 4, and in Fig. 8. One or more whole rings corresponding to these excisions are often seen in the polished cylindrical part of a punched hole. Here cracks, like that at s in stage 1, have started at various stages during the down-travel of the punch, each starting at the point where the sharp corner of the punch then lay. But their propagation has been arrested thus, and a little ring of triangular cross section has been torn out.

Now comparing conical with cylindrical punching, there is the difference that in the latter the strong pressure x, stage 1, opposing the propagation of the crack by holding together the metal transversely to the path of the crack, has no full equivalent at t', stage 5. This pressure x on the shoulder of the crack tends to bend that shoulder down, through a progressive change from the initially parallel sided crack, to angles less and less acute as in stage 3, and later even obtuse as in stage 4. Of these stages the parallel sided initial crack is seen in Fig. 5, A, Plate 1; stage 2 is seen in Fig. 5, B; and stage 4 in the wad of Fig. 8, about $\frac{7}{8}$ in. below t bottom of the plate.

A stage beween 3 and the obtuse-angled stage 4 is seen at the l edge of the plate in Fig. 8, on careful observation. Note that mn of the upper part of the projecting wad, though parallel to t the punched hole, is about $\frac{1}{4}$ in. at its left. If the punch were n farther down, the edge of the hole would not be at n but at q.

re-entering right angle *mnq* should be interpreted as simply a stage between stages 3 and 4, in which the gradually opening crack happens to have been caught while at a right angle.

Because the vertical component of the pressure transverse to the crack is so much feebler at t' in conical punching than at t in cylindrical punching, the crack at t' is able to extend; and it thus creeps upward till it reaches a region in which that transverse pressure is strong enough to arrest it, for instance till stage 6. As the punch moves lower and lower, and the thickness of metal still resisting its further motion decreases, the pressure due to that resistance decreases simultaneously, and with it the component of that pressure transverse to the path of the crack, and with that also decreases the resistance to the further propagation of the crack, which thus continues creeping upward till, as in stage 7, it reaches the cylindrical hole already formed. Hence the fact that the wad is conical throughout, lacking wholly any cylindrical part representing the cylindrical part of the hole.

But even in cylindrical punching there may be a short conical or bell part at the very bottom. The genesis of this is suggested in stage 4. At a stage intermediate between 3 and 4 the solid metal of the plate is once again pressed against the sharp corner of the die, and this starts a new crack quite as the first crack in stage 1 started. But by this time there is so little remaining thickness of metal resisting the punch, and the pressure which its resistance sets up is therefore so slight, that the component of that pressure transverse to the crack is no longer strong enough to arrest its propagation, and hence it creeps upward till it reaches the down-growing cylinder.

This intersection releases the last of the resistance to the forward movement of the punch, save the slight frictional resistance against the already cut sides of the hole, and hence releases the pressure within the metal of the plate, and hence the pressure on the top of the little bell r of stage 4, leaving it sharp and straight edged as at a in Fig. 4.

This individual bell a of Fig. 4 has thus cut through. But had it started somewhat earlier, while there was still enough pressure to force its upper wall down, that down-forcing might well carry that upper wall down past the corner of the die, giving rise to a thin plate-bottom burr, such as usually forms in cylindrical punching. A preliminary examination of these cylindrically punched holes indicates that part of the circumference of the lower edge of the hole may have such a bell, and the rest of it such a burr; but that burr and bell do not occur in the same place. Indeed, if we have sketched their genesis aright, they are mutually exclusive.

The intersection of the bell-shaped crack with the cylindrical hole might conceivably occur at the very corner of the punch, but it might also occur somewhat above that corner, as was the case in our conical

punchings, leaving the wad-top burr y of stage 7. In cylindrical punching a like but smaller wad-top burr forms in like manner.

The bending down of the wall of the crack in conical punching, because of the continuing vertical pressure on its shoulder after it has already opened, is indicated in Fig. 3. Note that the wall of the bell is not straight. At the bottom it is nearly vertical, but inclines more and more strongly to the right as we pass up toward the cylindrical part.

The reason why conical punching weakens a plate less than cylindrical punching,[3] in cases in which both are unpalliated by either reaming or annealing, is probably that it thus favors the propagation of the crack which starts at or near the corner of the die. At any given instant it is only the metal not yet cracked through that still opposes the downward movement of the punch. Hence the early and rapid propagation of the crack in conical punching reduces early and quickly the resistance to the entry of the punch. Moreover, the less resistance means less pressure within the plate, less friction between those parts of the plate already perforated and the punch which is moving through them, and less friction between the particles of ferrite and of pearlite which are dragged past each other from their initial equiaxed shape to the long zebra markings of Fig. 4.

In like manner the smaller pressure in conical punching, and the consequent smaller overstrain of the metal, might confine the damage to the metal in conical punching to a narrower ring.

These considerations furnish the explanation of the striking fact that though, with the narrow die, the distortion, as indicated by the length of the zebra markings, increases from top to bottom of the hole, with the wide die it decreases from the top of the bell down. Indeed at the lower end of the bell the arrangement of the ferrite and pearlite does not clearly disclose any distortion at all, and we need the presence of the heel-shaped burr at the bottom of the bell to assure us that there really has been material distortion.

Again, the decrease in the amount of deformation of the 2d order from top to bottom of the conical bell agrees with the conception that the crack which forms that bell starts at or near the edge of the die and creeps up, instead of starting at the edge of the punch and creeping down, because this deformation would naturally cease, or diminish very greatly, in any given region, at the moment when the crack extended beyond that region.

A comparison of these figures 3 and 4 shows us how much less reaming is needed to remove the distorted metal after conical than after cylindrical punching. If, for simplicity, the ratio of the diameter of reamer to punch is the same in both cases, then with the wide die the punch itself, in

[3] Howe: *The Metallurgy of Steel*, Table 122, p. 229.

tearing out the bell, removes most of the metal which must needs be reamed out if a narrow die is used.

An even closer measure of the extent of the deformed and embrittled region could probably be had by examining the difference of potential, on a surface cut tangentially to the punched hole, the cutting plane being parallel to the axis of the punch.

§5. The relative movements of the ferrite and pearlite masses in tensile rupture. Figs. 12 to 17, Plate 2, show the inequiaxing of these masses brought about in tearing a test piece in two in the common tensile test, both at the very edge of the fracture, and at a distance of $\frac{1}{2}$ in. from it. The micrographs are of course longitudinal sections. In Figs. 13, 15, and 17 the rough edge, with a wide plain black area below, is the fracture itself. Steels[4] of 0.21, 0.40, and 1.45 per cent. of carbon are represented. To Rosenhain's admirable exposition we have but a few points to add.

How closely the deformation is confined to the necking area in the case of low-carbon steels is seen on comparing Figs. 12 and 13 with Figs. 14 and 15, for whereas the inequiaxing, or drawing out of the ferrite and pearlite masses, at the edge of the fracture, is naturally much greater in the 0.21 than in the 0.40 carbon steel, yet at a distance of $\frac{1}{2}$ in. from the fracture the 0.40 carbon steel is rather more inequiaxed than the 0.21. Indeed one would hesitate to affirm that any inequiaxing of the latter is shown here. On the "delenda est Carthago" principle we point out that the elongation in the inch within which rupture occurs represents the capacity of the metal to undergo deformation without rupture, *i.e.*, its fitness for hyper-elastic uses, the equivalent of the contraction of area but more closely measurable; and that the elongation in the remainder of the test piece, excluding this inch, represents the ability to endure deformation without losing elastic strength, and hence the ductility available for hypo-elastic uses, including the great majority of engineering services.

Though such a steel, in this special structural state, *i. e.*, after such heat treatment as this has received, has only about 29 per cent. elongation in 4 in. in the common tensile test, the elongation of the individual ferrite and pearlite masses shown in Fig. 13 is several hundred per cent. A comparison of these masses with the long ferrite and pearlite hairs of Fig. 4 shows how the degree of deformation endured without rupture is increased by pressing the deforming masses firmly together. In this punching the elongation of the ferrite grains is not in the hundreds but in the thousands of per cents. Figs. 5, 6, and 8 of Plate 1 indicate that even the upper part of the wad does not actually break away from the

[4] These are the same steels which have served for many of our previously published experiments. The tensile test pieces were in each case $\frac{7}{16}$ or $\frac{3}{8}$ in. in diameter, with a measured length of 4 in.

sides of the hole till it has traveled far down that hole; that continuity persists thus through enormous local deformation.

This persistence is evidently due to the pressure with which each particle is held radially against its neighbor, contrasting strongly with the conditions in tensile testing, in which the withdrawal of each particle from its neighbor is not resisted externally in any comparable degree.

The slightness of the ductility of the steel of the 1.45 carbon, when its pro-eutectoid cementite has thus harmfully been assembled into a network, is reflected by the almost complete lack of inequiaxing in this network, even at the very edge of the fracture.

Fig. 16 illustrates the grave damage done by graphitizing the pro-eutectoid cementite, when this has been drawn out into long ghosts by rolling. The little equiaxed craters scattered through this field are the common ferrite-edged graphite masses resulting from the graphitization of cementite. The long white streak is, in like manner, ferrite which has resulted from this graphitization of a cementite ghost, and the black band with the white edges is a graphite mass, which is in effect a very formidable internal crack.

It is indeed striking that these cementite ghosts should be present in spite of the abundant opportunity for the diffusion of the carbon, for since this steel was rolled it has had two long sojourns well above the transformation range, one of an hour at 1,100°, and another of an hour at 1,000°.

§6. The shattering of coalesced cementite during plastic deformation is shown in Figs. 18 and 19. The long Saghalien-shaped vertical island of cementite, which is now shattered transversely into five fragments, was evidently a cementite reef formed by pearlite divorce. Its smooth upper surface distinguishes it clearly from the etch-figured sea of ferrite which bathes it. Whether or not all the crevices between its sharp fragments have been completely closed and welded by the inflow of ferrite or not, it is clear that, at the moment when any one of these transverse ruptures occurred, a minute crack must have formed. The ferrite evidently has a better chance to fill and heal these gashes, to weld across them, and to form strong adhesion with the cementite, under the slow deformation of the tensile test than in the sudden deformation of the notched bar impact test, for time certainly ought to favor these healing processes. Here, then, we have the justification of the assertion that the specific kind of brittleness, caused by the divorcing annealing of this low-carbon steel, is disclosed by the impact test more clearly than by the tensile test.[5]

§7. Intra-pearlitic deformation. This deformation may be recog-

[5] Howe: *Trans.*, xlvii, 506 (1913); The Authors: *Transactions of the Cleveland Institution of Engineers*, to appear.

nized best by proceeding at once to the extreme case of punching, illustrated by Figs. 21 and 22, in each of which the direction of punching is vertical, as is shown by the zebra markings. Note the mass of shattered pearlite shaped like a pear, with its tail pointing upward. In the right-hand part of this mass, at *a*, the stratification of the pearlite is parallel to the direction of punching, and here the flow has elongated the pearlite parallel to its stratification. This elongation has narrowed the ferrite lamellæ proportionally, but the cementite lamellæ, being inextensible, have broken up into fragments which preserve roughly their initial direction.

As this thinning of the ferrite lamellæ suggests the closing of an accordion, so there is much to suggest its opening, with the moving apart

FIG. 31.—IF ALTERNATE STRATA ARE BRITTLE, COMPRESSION PARALLEL TO THE STRATIFICATION CONVERTS "CLOSE ORDER IN LINE" INTO "OPEN ORDER EN ECHELON."

of the cementite lamellæ by the foreshortening of the ferrite lamellæ at *b* in the lower part of this pear, where the pearlite stratification is normal to the direction of punching. The foreshortening of the plastic ferrite lamellæ is easily understood, but it is so hard to understand either how the brittle cementite lamellæ can thus foreshorten, or how they can, without breaking, endure the flow of the ferrite lamellæ between them and along their faces, that we should receive this suggestion with caution.

One may be less skeptical toward the like suggestions of the opening of the accordion, and the implied foreshortening of the ferrite lamellæ, in Fig. 19, because in the lower part of this pearlite mass several of the cementite lamellæ lie *en echelon*.

This suggests that the widening of the ferrite "lines," speaking militarily, has tended to move the cementite "lines" into more "open order," but that the shortening of the lines which this widening implies, though it is readily obeyed by the plastic ferrite, could not be followed directly by the rigid cementite, which instead has broken into "squads" in "*en echelon*" order, as sketched in Fig. 31.

This arrangement *en echelon* is seen also in the lower part of Fig. 19, a longitudinal section close to the fracture of a tensile test piece of 0.21 carbon steel. Because this area is close to the edge of the punch, it has bent down like a snow cornice. Though this bending gives the ferrite lamellæ nearest the edge a look of having been narrowed, so that the cementite lamellæ are brought closer together, yet at a distance of $1\frac{1}{2}$ in. from the edge of the hole on this magnification it seems to have

opened the accordion, *i.e.*, to have increased the distance from cementite lamella to lamella, implying a thickening of the ferrite lamellæ between them, and with that thickening a foreshortening, which tallies well with this *en echelon* arrangement.

What is even more striking in both of these figures is the apparent bending of the cementite lamellæ without breaking. The curving and continuous change of direction of the pearlite stratification from *a* to *b* of Fig. 21 is quite foreign to the natural habit of pearlite, and seems due to some eddy movement of the whole region. Much sharper curvature of the strata is seen in the upper part of Fig. 19, where the right-to-left compression has crumpled the stratification up into a W shape which is certainly artificial. Though one cannnot be sure that these individual cementite lamellæ have not broken in thus distorting, it is certain that some of these lamellæ in Fig. 19 have bent almost to a U without breaking. A striking example is the lamella marked *c*.

That these extremely thin lamellæ of cementite should thus bend without breaking, though the thick cementite island of Fig. 18 has broken, is natural, reminding us of the great flexibility of sufficiently thin threads of slag-wool or glass, substances which in thick masses are so inflexible. Curving of the pearlite through flow is suggested by the left-hand part of Rosenhain's Fig. 19,[6] but without clear indication as to whether the cementite lamellæ have remained unbroken through this bending.

Fig. 22 shows what we have called a "laccolite," *i.e.*, an intrusion of pearlite into the strata of ferrite with which it was initially interstratified. If not strictly a laccolite, it is like true laccolites in being arrested and not breaking through to the surface. Here the transverse component of the flow has forced an intrusion of pearlite into three parallel layers of ferrite, and indeed clear through the left-hand layer. What has become of the ferrite displaced by the intrusion of pearlite into the right-hand ferrite stratum is not clear.

Another unusual form of pearlite, cellular and recalling lebeburite, is shown in Fig. 7, Plate 1. Here the orientation of the etch figuring of the ferrite is not only very constant, but alike in these two ferrite bands, as if it were due to the pressure or the movements in punching.[7]

Disruption of the pearlitic cementite in punching, by translation transverse to its stratification, is shown in Fig. 26. Though the general flow in the extra-circumferential metal is radial and away from the punch, *i.e.*, tending to narrow any strata which lie parallel to a plane tangent to the hole, yet such strata, when they lie in immediate contact with the punch, are dragged down by it, curving like the surface of water at the edge of a fall, so that their stratification is widened, *i.e.*, the cementite

[6] *Journal of the Iron and Steel Institute*, 1906, No. 2, Pl. XLVI, following p. 224.

[7] These markings may possibly be connected with the polishing of the specimen.

strata tend to move apart. In Fig. 26 the mobile ferrite has conformed to this local opening of the accordion, but the brittle cementite lamellæ have broken up into short fragments. Here the surface is so much curved that only a very narrow strip of it can be brought into focus at once.

In like manner Fig. 24, the longitudinal section through the tensile fracture of a 1.45 carbon steel, shows in its left-hand part an apparent opening up of the stratification, which lies transversely to the axis of the test piece and the direction of main flow, whereas in Fig. 23, also a longitudinal section of the same fracture, the pearlite, which is here stratified nearly parallel to the flow, is but little deformed.

Another case of the deformation of the pearlite in tensile rupture is shown in Fig. 20, a longitudinal section near the fracture of a tensile test piece of 0.21 per cent. of carbon, with an unusual pearlitic structure.

The intra-pearlitic movement during the wire-drawing of bridge-cable wire, of about 0.75 carbon, is shown in Figs. 28, 29, and 30. This rod, which was $\frac{1}{4}$ in. in diameter, was drawn part way through a die $\frac{3}{16}$ in. in diameter, the rear of the rod remaining unreduced. Fig. 28 shows the unreduced rod, and Figs. 29 and 30, which represent spots within an inch of Fig. 28 in this same specimen, are from the wire into which this part of the rod has been drawn. Fig. 28 shows the normal pearlite of the rod, Fig. 30 shows about the normal condition of the pearlite in the drawn wire, and Fig. 29 shows one of the least disturbed pearlite areas in that wire.

Though the pearlitic deformation is very marked in some of these cases, yet in others it is only slight even when the presumable deformation of the 1st and 2d orders is very considerable, indicating that the movement of the grains past each other, and the readjustment of the positions of the masses of ferrite and of cementite, may leave the stratification of the pearlite little deformed This can be traced in the right-hand part of Rosenhain's micrograph, No. 19, just referred to. Here the deformation of the pearlite at the very edge of a tensile fracture is at most very slight.

The smaller general deformation of high- than of low-carbon steel would naturally create a smaller tendency to intra-pearlitic deformation, yet Fig. 24, at the edge of a fracture of a very high-carbon steel with 1.45 per cent. carbon, indicates that even in this the pearlite itself may open up much. This makes it all the more striking that the pearlite of Fig. 25 shows so little deformation, even under the abrupt deformation of the mass which should be expected here at the very edge of a punched hole. Other cases, Figs. 23 and 29, of slight intra-pearlitic deformation where the deformation of the second order is very great, have already been noted.

§8. Equiaxing Phenomena. The contrast between the long ferrite ghosts in the left part, *L*, of Fig. 27 and the equiaxed ferrite network in

its right-hand part, *R*, is very suggestive. *L* is evidently much decarburized, so that the temperature of Ar3 for *L* is higher than for *R*. From these facts we may infer that when the rolling ended the temperature of *R*, which was absolutely higher than that of *L* because of the greater distance from the surface, was far enough above the Ar3 of *R* to permit the austenite grains to recover from the distortion caused by the rolling and to equiaxe. Further, it suggests that the temperature of *L* was within its transformation range, *i.e.*, below its Ar3 but above Ar1.

That it was below Ar3 is suggested by the great elongation of the white ferrite ghosts. For above A3 the re-equiaxing of the distorted austenite grains seems to be rapid, so that if the finishing temperature had been above Ar3 these austenite grains would have equiaxed, and the ferrite which they generated in cooling from Ar3 toward Ar1 would have formed an equiaxed network. Yet this indication is not decisive, because such ferrite streaks form even after apparently good opportunity for the equiaxing of the austenite.

That this finishing temperature was above Ar1 is indicated by the presence of some equiaxed cells at the extreme left. An equiaxed ferrite boundary can be formed only by expulsion from an equiaxed grain; so that these equiaxed boundaries consist of ferrite expelled from grains which at the time of that expulsion were equiaxed, and have remained so since, *i.e.*, there has been no rolling since that expulsion. But ferrite expulsion comes only between Ar3 and Ar1. Hence the rolling here must have ended above Ar1.

§9. Acknowledgment. This investigation has been carried out in part under a grant from the Carnegie Institution of Washington.

§10. Summary. Four aspects of flow are distinguished, the movement (1) of individual grains past each other, (2) of individual masses of pearlite and ferrite relatively to each other, (3) of the pearlitic ferrite and cementite within the pearlite, and (4) of the crystal units along slip planes. (§1, p. 532.)

2. The inter-granular movements as shown on the previously polished upper surface of a punched piece are more severe, but apparently confined to a narrower ring on that surface, in cylindrical than in conical punching. (§2, p. 533.)

3. A study of the movements of the individual ferrite and pearlite masses during punching suggests a tentative explanation of the differences observed between cylindrical and conical punching. (§§3 and 4, pp. 533, 535.)

4. The corresponding movements in tensile rupture are examined. (§5, p. 540.)

5. The shattering in tensile rupture of coalesced cementite islands, arising from the divorce of the pearlite in low-carbon steel, is illustrated. (§6, p. 541.)

Table I.—Chemical Composition and Tensile Properties of the Steels used in These Experiments

No.		No. of piece	Chemical composition					Tensile properties				Shown in figure No.
			C	Si	Mn	P	S	Tensile strength	Yield point	Elongation in 4 in.	Contraction of area.	
			%	%	%	%	%	lb. per sq in.	lb. per sq. in.	%	%	
1	½-in. tank plate....	T2, T3.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	9, 10, 11
2	⅝-in. round bar.....	IV, 7.....	0.21	0.039	0.053	0.013	0.010	57,385	33,790	29.0	61.9	12, 13, 18, 19, 20
3	⅝-in. round bar.....	I, 1.....	0.40	0.103	0.161	0.014	0.012	69,900	22,480	23.01	36.7	14, 15
4	½-in. plate.....	I, 251; I, 252	0.40	0.103	0.161	0.014	0.012	.....	.....	.....	.....	3, 4, 21, 22
	½-in. plate.....	I, A, B, C, D	0.40	0.103	0.161	0.014	0.012	.....	.....	.....	.....	5, 6, 7, 8
5	½-in. wire rod.....	205.....	0.75±	.....	.....	.....	.....	.....	.....	.....	.....	27, 28, 29, 30
6	⅜-in. spring plate...	C1.....	0.78	0.108	0.26	0.011	0.013	.....	.....	.....	.....	25, 26
7	⅝-in. round bar.....	IX, 8.....	1.45	0.14	0.16	0.009	0.006	89,250	35,780	3.8	2.91	16, 17, 23, 24

Table II.—Details of the Punching of the Steel Plates Represented in Plate 1

Cylindrical punching

	C %	Loss of volume of wad in punching *a*		Lengthwise compression of wad, per cent. of initial thickness of plate	Shortage of diameter of wad, in per cent. of diameter of hole		Approximate clearance between maximum circumference of wad and		Approximate clearance between inside diameter of hole and die	
		By measure	By weight *b*		Top	Bottom	Die (inch)	Punched hole (inch)	Inches	
Tank plate (average of three holes).	low	4.7	10.2	0.4	1.2	1.2 ±*d*	0.0065	0.0015	0.005	½-in. plunger
I, 251, C.....	0.40	4.6	14.5	2.4	1.2	1.2 ±	0.0065	0.0015	0.005	½-in. (0.26) die.

Conical punching

I, 252, D.....	0.40	10.3	4.3	9.7	−2.0 *c*	7.0	0.025	0.0125	0.0125	½-in. plunger.
I, 252, E.....	0.40	10.0	9.1	6.5	0	8.3	0.0275	0.015	0.0125	
I, 252, F.....	0.40	11.7	3.7	9.7	−2.0	9.1	0.0265	0.0165	0.010	⅝-in. (0.385) die.

a Reckoning volume of holes as 100.

b Taking specific gravity as 7.85.

c The top of the wad is 2 per cent. wider than the top of the hole.

d Not determined; the hole and wad are nearly cylindrical.

6. The intra-pearlitic deformation is studied. Elongation and compression parallel to the pearlite stratification lead respectively to drawing out the cementite lines into squads, and rearranging them *en echelon*. These cementite lamellæ sometimes curve greatly without breaking (§7, p. 541). A pearlite "laccolite" eruption through the ferrite layers, with which it was initially interstratified, is shown (p. 543). Though the pearlite stratification is often greatly crumpled and otherwise deformed, yet it sometimes remains little deformed at the very edge of a fracture (p. 544). Even in high-carbon steel this pearlitic deformation may be marked (p. 544).

7. Incidentally it is shown how the equiaxing phenomena may be interpreted. (§8, p. 544.)

Discussion

H. C. H. Carpenter, London, England.—There are one or two points of perhaps general interest on which I should like to touch. I notice that Professor Howe refers to the work of Ewing and Rosenhain in connection with the plastic deformation of metals. That, of course, is quite a classical work. Their theory was that in that formation you get the metal slipping along its gliding planes in a series of steps. Now, it is very curious that that work, although quite right up to a certain point, has been comparatively little used, and, indeed, Professor Howe himself does not make any very great use of it in this paper.

One of the main reasons why the work has not been more used, I believe, is this: Professor Ewing and Dr. Rosenhain showed, in the plastic deformation of metals with which they experimented, that these undoubtedly began to slip by a series of steps, but they were not able to give a reason showing why, when once a metal has started to deform, it should ever stop deforming until it was broken.

Now, of course, we know that it does; the metal can be deformed up to a certain point with a given stress, and then, after that, the stress has to be increased if any further deformation is to take place. No explanation of that increased stress was given by the authors.

That brings me to the work of another researcher in our country, Dr. Beilby, whose work is not so well known as it deserves to be, particularly because his work has been overlooked, for the reason that he chose metals which are simpler than iron—gold, silver, and copper, which can be easily obtained in a state of purity, and, therefore, the results can be interpreted more easily. Dr. Beilby furnished an interesting interpretation of where these begin and end. Dr. Beilby showed, and I think conclusively, that in addition to the actual slip by steps, or under these gliding planes, you get an actual transformation from the crystalline solid metal into what is virtually a liquid—what are called amorphous vitreous phases. This phase is brittle, and what happens is that when

the stress is applied, and the metal begins to move along the gliding planes, then the rubbing of the crystal surfaces on one another produces the vitreous brittle phases, and ultimately the crystal units which are broken up are incased in an enamel of this vitreous material, and after that rupture can only take place by breaking across these vitreous phases. All of this is very fully indicated and brought out in Dr. Beilby's paper published in 1910 or 1911. I imagine that Dr. Howe is familiar with Dr. Beilby's work, as he quoted his name in connection with the paper, but it is surprising that more use has not been made of Dr. Beilby's work in this connection.

I should like to ask the authors whether they had any difficulty in developing the structure of their pearlite and other steels by etching. The effect, of course, of the cutting work is to give us ultimately this mixture of crystalline and vitreous materials, this etching, rather differently.

Some years ago I was consulted by a firm which was having difficulty with the stamping of steel links for chains. The carbon percentage of these links was about 0.01 per cent., and I found great difficulty in developing the structure of that steel in the neighborhood of the punch. That again, I think, is explained by the theory of the amorphous vitreous phases which are formed in the neighborhood of the fracture. We will have, undoubtedly, different properties in the etched material from the crystalline material.

In this paper there is a certain amount of evidence. Turning to Figs. 3 and 4, I notice that on the extreme right we have an area which is apparently hardly etched at all. I do not know whether that is because the material is decarburized.

Professor Howe.—It is decarburized.

H. C. H. Carpenter.—That is a point I wanted to be clear about. I do find that difficulty with regard to etching, and I found I was only able to get comparable results by giving these severely strained links a low-temperature annealing in order to reconvert that amorphous phase into the crystalline phase. That has to be done at a temperature below the melting conversion, but I found a temperature of 500° to 550° was insufficient.

I would like to ask a question of the author. These photographs, as I gather, are of materials just in the strained conditions, without any other treatment. I would like to ask if they have had any difficulties in interpreting their stresses? I should like to express, in conclusion, my great admiration for the paper by Professor Howe and Mr. Levy.

Henry D. Hibbard, Plainfield, N. J.—These views showing the surface of the distorted steel remind me of some compression tests of manganese steel I made a great many years ago, the test pieces being

cylinders 1 in. in diameter and 4 in. long. They were cast test pieces, ground to dimensions. After compression it was possible from the appearance of the surface, which looked much like these photographs, to tell about the coarseness of the crystallization or granulation of the metal within, the test piece having been shortened so that its diameter was increased by bulging perhaps a fourth.

William R. Webster, Philadelphia, Pa.—Several years ago, in order to show the injurious effect on steel by punching and shearing, even after the rough edges had been removed by reaming or planing, I took some pieces of steel plate and punched two lines of parallel holes in them about 3 in. apart, then cut out the longitudinal strips between these punched holes. These strips were pulled in the tension testing machine and in all cases showed hard spots opposite the punched holes, and at other points, where the metal had not been hardened by punching, it necked down. In all cases the fractures occurred at one of these points where the metal had not been hardened, but the hardening effect of the punching extended in much further than I had expected.

Ralph H. Sweetser, Easton, Pa.—I would ask Dr. Howe if temperature has anything to do with the final rupture of the steel, if it is under stress; that is, will intense cold help the final rupture, when the shock comes, or will the higher temperature, say of superheated steam, have anything to do with the rupture of steel under stress?

C. P. Linville, South Amboy, N. J.—I should like to ask Dr. Howe whether the surface hardening caused by wire drawing, cold rolling, and such processes may not to a certain extent be explained by this paper. It would seem to me that in the case of steels having some cementite, even though it be pearlitic, such surface treatment would tend to drive, or cause the softer ferrite to flow away from the surface, thus leaving a shell much richer in cementite and accordingly harder. Another instance which I have observed is the case of rails that have been a considerable length of time in service, so that the top has been worn down. I have found a thin skin, extremely hard, which was apparently altogether cementite. On a polished section the skin would not etch when treated with acid and under the microscope was white and in its behavior like cementite. It was also noted that in this skin numerous small surface cracks were visible, which might eventually be extended deep enough into the rail to cause weakness. I realize that there is a certain amount of surface hardening, even in pure metals, which has been explained as being due to the formation of an amorphous layer, but it seems to me that in the case of the carbon steels, the driving out of the soft ferrite, leaving the hard cementite on the surface, may also have an important effect.

Henry M. Howe.—I am sure it is a great gratification to us to listen to the successor of Dr. John Percy and Sir William Roberts-Austen. Dr.

Beilby's wonderful theory of a temporary liquid state between the natural crystalline state and the final amorphous state, such as we find in hard-drawn and cold-worked metals generally, certainly does explain an enormous amount of phenomena that before that was perfectly unknown and not understood. Why sliding should start a step, of course, was very difficult to understand. If the thing started to break, why did not it keep on breaking; if the stress was able to start, what should stop the started crack?

Dr. Beilby's theory does explain that in the most satisfactory way, so far as the phenomena of simple overstrain is concerned, but it seems to me that it breaks down at the phenomena of reversing overstrain; that is, if you strain a piece of metal past the elastic limit in tension, and reverse the strain past the elastic limit in compression, the thing seems so far to have brought about a set of results which were very different from those you got on simple overstrain.

The hardening effect of simple overstrain cannot be due to bringing the cementite to the surface before the development of the amorphous state, because that is the case in pure metals—pure copper is hardened the same way by overstrain—it hardens pure metal just the same as it hardens impure ones. The purity of the metal does not decrease the degree of hardening at all, so that it is a perfectly satisfactory explanation, but the generation of the harder amorphous metal, why that metal should be harder, is extremely reasonable and natural when you consider the natural conditions of the case.

If simple overstrain makes the metal lighter, harder, etc., the tests made so far do not indicate that reversing overstrain either hardens the metal or lightens it. Effects of simple overstrain may be removed absolutely and completely by heating to 600° C., or, as Professor Carpenter says, an even lower temperature. The effects of reversing strain do not seem to be the same. The evidence so far as it goes does not indicate clearly that reversing overstrain is removed by such heating, but it is only fair to say that the bulk of the annealings which have been made were not carried to a high enough temperature to establish that point. That is a point which is still under debate.

The tensile strength, depending on the temperature, certainly rises with the temperature up to a moderate degree. There is a very steep rise of the tensile strength curve with rising temperature, followed by a drop again, if you get to about 200° C. There is a reversal, but from that temperature down, as the temperature falls, the tensile strength diminishes. And that may be bound up with this phenomena of the play between the cementite and the ferrite during deformation. That, however, I do not think we can feel sure about yet.

Professor Henry Le Chatelier, Paris, France (communication to the Secretary*).—I have read with interest the paper of Messrs. H. M.

*Received Mar. 26, 1914.

Howe and A. Levy on the plastic deformation of steel, and was particularly struck by the observations concerning the cementite lamellæ of the pearlite, which were bent without fracture. Must they not have been, then, in elastic tension? Their solubility in ferrite would be increased as a consequence. If then a steel in which the cementite is thus deformed be heated to a temperature near the point of recalescence, the curved lamellæ ought to disappear and recrystallize out; but under what form? As lamellæ or as rounded grains? A crystal soluble in water, if subjected to mechanical stress has its solubility increased. The same is necessarily true in metals. This may explain in part the facility with which metals partially hardened, and thus out of equilibrium, can dissolve and recrystallize in the presence of very small quantities of solvents.

Investigations of Coal-Dust Explosions

BY GEORGE S. RICE, PITTSBURGH, PA.

(Pittsburgh Meeting, October, 1914)

THE subject of dust explosions in coal mines first appears in the *Transactions* of this Institute following the first great mine disaster that happened in bituminous mines of the United States. This was the Pocahontas explosion, which occurred Mar. 13, 1884, causing the loss of 112 men. The operating company wisely asked the technical assistance of the Institute, requesting that a committee be appointed to investigate and report. This committee consisted of Messrs. Bramwell, Buck, and Williams, who, after careful study, reached the conclusion that the explosion was due "either to dust alone or to dust quickened by an admixture of fire-damp too slight for detection by ordinary means." This is perhaps the first time in this country that coal dust was considered to be the chief agency in a mine explosion. The report is given in the *Transactions*, vol. xiii, p. 237 (1885). It is particularly worthy of note that the committee called attention to "the necessity of full-sized tests and systematic experiments."

In the same volume of the *Transactions*, there is a most interesting paper reviewing the literature relating to coal dust, by E. S. Hutchinson, entitled Notes on Coal-Dust in Colliery Explosions. Mr. Hutchinson gives a historical account of observations on the part played by coal dust in mine explosions, beginning with Messrs. Lyell and Faraday's report on the Haswell colliery explosion in 1844, in which it is said "much coal gas had been made from this dust in the very air itself in the mine by the flame of the firedamp, which raised and swept it along, and much of the carbon of the dust remained unburned only from want of air."

The subject lay dormant until 1875 when William Galloway began some coal-dust experiments in a small gallery at a mine in South Wales of which he was manager. At that time he considered the presence of firedamp necessary for continued propagation of a dust explosion; but in 1884, the year of the Pocahontas disaster, he stated that no earlier author than himself had credited coal dust with being a principal factor in mine explosions, relegating firedamp to a secondary place.

Meantime in 1876, Henry Hall and George Clark made some coal-dust

explosion experiments in an adit 45 yd. in length, which were the first large-scale tests undertaken with coal dust. The explosions traveled the length of the adit, and issued with violence from the mouth.

In 1878 Morison and Marreco began experiments in diminutive galleries, followed in 1880 by similar experiments conducted by Professor (afterward Sir) Frederick Abel, with a view to determining the cause of the explosion disaster that occurred at the Seaham colliery. Professor Able drew the conclusion that as far as could be determined by experiments on a moderate scale the tendency of coal dust by itself to propagate flame is of a limited nature, and it required the presence of a very small percentage of firedamp; although he admitted that in the complete absence of firedamp, an explosion might be propagated to a greater distance than the results of small experiments would warrant one in assuming.

Mr. Hutchinson, in his paper, summarizes the opinions of the inspectors and prominent mining men before a Royal Commission sitting in 1879 and 1880. All those whose testimony he cited "appeared to be in accord that while dust in itself is not explosive, it is abundantly able to intensify an explosion initiated by gas." He then cites at length a series of 211 explosion tests conducted by the Chesterfield and Derbyshire Institute of Engineers, under the direction of Morison and Marreco, in a wooden tube 82 ft. long and 14 by 18 in. in section on the inside. It is interesting to note that in only a small proportion of the tests was ignition obtained and apparently in no case did propagation extend throughout the gallery, so that the conclusions of the committee were unanimous that "only *ignition* of dust (in distinction to an explosion) occurred *without* gas," and they added: "special circumstances must combine to insure ignition of dust *without* gas."

At the conclusion of Mr. Hutchinson's paper he says: "Neither assent nor dissent with regard to the views that have been quoted is intended to be expressed in this paper. But the facts, nevertheless, remain that there are dusts and *dusts*, and that the ratio of the number of serious mine explosions to the total number of blown-out or overcharged shots and local explosions of fire-damp, in dry and dusty mines, is a small one; whereas, if the coal-dust theory be true to the extent insisted upon by its most strenuous advocates, every such local occurrence, in a mine of this description, should be followed by the more or less complete wreck of the colliery." This indicates the general point of view in 1884.

The reason for the many failures to obtain explosions of coal dust in the earlier experimental galleries was perhaps due to one or more of these causes:

1. Degree of impurity of the coal dust used.
2. Large size of the particles of dust.

3. Smallness of igniting source, that is, the flame too short in length or in duration, or, the mechanical energy insufficient to raise the dust into a cloud.

4. Taking into account the small size of the igniting charge, it is possible there was unsuitable placing of the dust, or an insufficient quantity of the dust at the point of ignition.

The next discussion of coal dust in mine explosions, which appeared in the *Transactions*, in vol. xxiv (1894), followed a paper by William Glenn on Mine Explosions Generated by Grahamite Dust, in which he evidently considers in the two explosions in 1871 and 1873 in the Ritchie mine (West Virginia) that the dust was the cause and not firedamp. In the discussion Dr. R. W. Raymond quoted at length from the report of the Prussian Fire Damp Commission which, in 1884, began a series of experiments with coal dust, and coal dust and gas, in a gallery 167 ft. long, elliptical in cross-section, the diameters of which were 4 ft. (horizontally) and $5\frac{1}{2}$ ft. (vertically). As was stated by Dr. Raymond, these experiments were more systematic and thorough than any previously conducted.

The general conclusion of the Commission was that when firedamp is absent the propagation of an explosion of coal dust is limited for most varieties of dusts, but there are coal dusts which once ignited by a shot, burn, or spontaneously give flame far beyond the localities strewn with the dust.

In Austria in 1885 and 1886 coal-dust experiments were conducted in a testing gallery. In these experiments nearly all kinds of coal dust were ignited by a cartridge of 100 g. of loose dynamite, but it was stated that a small mixture of firedamp notably increased the sensitivenesss of coal dust to ignition. Further, that the fineness of the dust, as well as its dryness, greatly increases its sensitiveness and danger.

In 1886 W. N. and J. B. Atkinson, mine inspectors of England, issued an interesting book entitled *Explosions in Mines* which was the result of their studies of many mine explosions. The authors state very positively that they consider coal dust the chief factor in most of the British mine explosions of the past, and describe in detail six explosions, in five of which they consider coal dust was the chief, if not the only, agency. Their discussion of the subject shows remarkable knowledge of the new problem. They considered the most dangerous dust is the finest coal dust, or "float dust," which they term "upper dust" from its being found on the timbers and on projections of the roof and sides. In the light of the definite information obtained in recent large-scale experiments the authors possessed a remarkable intuition of the mechanics of explosions, the circumstances under which coal-dust explosions will propagate, the position of the coked dust with reference to the movement of the explosion, and many other details, which shows that their views were well in advance of the times. It is particularly interesting to note that they concluded

"that the direction that the air was traveling had no influence on the course of an explosion after it was once started."

In 1890 to 1893, Henry Hall, British Inspector of Mines, conducted some explosion tests in disused shafts, which led him to state that "his experiments conclusively prove that a blast of gun powder in dry and dusty mines may cause serious disaster *in the entire absence of firedamp.*"

The British Royal Commission, in 1894, in its summary states: "Coal dust alone without the presence of any gas at all may cause a dangerous explosion if ignited by a blown-out shot or violent inflammation. To produce such a result, however, the conditions must be exceptional, and are only likely to be produced on rare occasions."

The matter rested for many years following this report, not only in this country but abroad, except for occasional discussions. Nothing on the subject of coal dust in relation to mine explosions appears in the *Transactions* between 1895 and 1908, when there was a paper presented by Carl Scholz, entitled Effect of Humidity on Mine Explosions, which brought forward the danger of dry coal dust in mines, and discussed the advantages of the sprinkler-humidifying method of preventing mine explosions (vol. xxxix, p. 328 (1908)). This was followed by a series of papers on these and allied matters by Bache, Shurick, Mannakee, Raymond, and Rice, appearing in vols. xl and xli. Meantime, however, it had become apparent that long-flame explosives were dangerous to use in dusty and gaseous mines, and galleries were established in Germany in 1894 and subsequently in Belgium, Austria, and England for testing so-called "safety explosives," which through giving a shorter flame, one of less temperature, and of far less duration, would be relatively safer than black powder or dynamite. This led to the establishment of "permitted lists" of explosives in Belgium and Great Britain, and specifications of requirements in France, Germany, and Austria.

It was not until after the great Courrières disaster in 1906, which cost the lives of 1,100 men, in a colliery which had always been so remarkably free from firedamp up to that time that it was considered non-gaseous, that the fear of coal dust was reawakened in the great coal-mining countries. It led in France to the establishment by the Central Committee of Operators, of the Lievin station, which was placed under the talented government engineer, J. Taffanel. Prior to this French mining men had been led to believe, by early laboratory tests, that firedamp was the chief danger, and that coal dust would only increase the intensity of an explosion, and therefore if ignition of firedamp was prevented there would be little danger from coal dust itself.

In Great Britain the Coal Operators Association, early in 1908, erected the large Altofts gallery, which was placed under the direction of Mr. (later Sir) William E. Garforth, who had repeatedly called attention to the danger of coal dust, and who believed, as a result of his observations following the

disaster at the Altofts collieries in 1886, that rock dust, if present in sufficient quantities, rendered coal dust non-inflammable. The important work at Altofts continued for three years, and experiments were conducted on a larger scale than had yet been tried, the gallery having a total length of about 900 ft., the explosion section proper being 600 ft. long and 7½ ft. in diameter. Many new and valuable data were obtained from these experiments regarding the propagation of dust explosions, the force developed within the limits of the gallery, the gases formed and the effectiveness of the rock-dust treatment method of prevention.[1] In 1911 this work was transferred by the Mine Owners Association to the British government and the gallery was removed to Eskmeals, where tests have since been conducted under the direction of a Royal Commission and five brief reports issued.

The Lievin station in the Pas de Calais district of France, when first established in 1907 employed a small gallery for preliminary tests, but a larger gallery was installed in 1908, and successively increased in length until the scale on which the tests were conducted was more extensive than in the British experiments, the main gallery being 1,000 ft. in length. Mr. Taffanel has advanced the knowledge concerning coal-dust explosions to an extent which places the work on a more exact scientific basis, particularly as regards the explosion wave movements under different conditions. The acquirement of information has, in part, been due to the high development of the recording instruments he employs.[2]

As already stated, explosives testing galleries were established 20 years ago in Germany and subsequently in Belgium and Austria. In these galleries tests of explosives are made in the presence of gas and dust, and some dust-explosion experiments have been conducted, but the galleries are too short to enable conclusions to be drawn regarding the wave movements of explosions except to a limited extent in the underground gallery at Rossitz, Austria. The new Westphalian surface gallery near Dortmund, Germany, now 100 m. long, it is intended will be enlarged and in the future experiments will doubtless be conducted of a size and thoroughness comparable with those being carried on at Eskmeals and Lievin.

Coal-Dust Experiments Conducted by the Bureau of Mines

The coal-mining industry of the United States, as compared with that industry in European countries, was very backward in taking up experimental investigations relating to coal dust, although mine explosions attributed to coal dust had happened with sad frequency. Just why more active interest was not taken in experimental investigations is not ap-

[1] See Record of the First Series of the British Coal-Dust Experiments, 1910.

[2] See series of publications issued by the Comité Central des Houillères de France, 1909 to date.

parent, because the magnitude of the industry in this country fully warranted such investigations. Neither the individual operators or associations nor States took the matter up, although Pennsylvania alone mines more coal annually than any foreign country except Great Britain and Germany. Finally, however, in 1907, such a series of disastrous mine explosions occurred that the country was aroused, and Congress appropriated money for the investigation of mine explosions and other mine accidents, which subsequently led to the formation of the Bureau of Mines. Thus the first, and so far, the only large-scale coal-dust experiments conducted in this country were started by the Federal government at the Pittsburgh station, established in the fall of 1908. During the first year the Pittsburgh gallery, a 100-ft. steel tube, $6\frac{1}{2}$ ft. in diameter, was devoted to the testing of the so-called "safety explosives" in the presence of gas and dust, to establish a permissible list of those that passed the tests; but in the following year a preliminary series of experiments was conducted in the gallery to obtain information concerning the explosibility of coal dust, and how the danger might be neutralized. Information on the following points was sought:

1. To determine how small a quantity of coal dust will propagate an explosion.

2. To find the coarsest sizes of dust that will propagate an explosion.

3. To determine how much moisture coal dust must contain to prevent ignition and propagation.

4. To observe the effect of foreign inert matter or ash upon ignition and propagation of an explosion.

These tests were largely conducted with dust ground from coal of the Pittsburgh bed, which is considered to produce dust of the most inflammable character. It was found that, when very fine (through 200 mesh), as small a quantity of dust as 32 g. per cubic meter of space (0.032 oz. per cubic foot) would ignite and propagate, at least within the short limits of the explosive zone employed. (See *Bulletin No.* 20, *U. S. Bureau of Mines,* Explosibility of Dust.)

As regards the second inquiry, it was found that with coal screened through a 20-mesh screen and over 40 mesh, ignition and partial propagation would ensue, and as it had been found that with every increase in the igniting charge, up to the largest amount that could be put in the cannon, there was an increasing ease of ignition, it seemed probable that a larger charge of powder in the initiating shot would cause complete propagation of dust between 20 and 40 mesh. It was therefore tentatively decided to accept 20 mesh as the dividing line between small-sized coal and what might be considered dust. While this limit is empiric, it is found to be a good division in practice, and has since been adhered to in gathering samples of road dusts; if larger sizes are gathered, that part over 20 mesh is rejected.

In the third inquiry, as to the effect of humidity and moisture, it was found that the humidity of the atmosphere itself had no appreciable effect upon an explosion of inflammable dust, but when the moisture content of the dust was such as to prevent its being thrown in suspension, then no explosion resulted. These features have since received abundant verification by numerous tests at the experimental mine.

As regards the fourth principal inquiry, only a few tests were conducted to determine the effect of the ash or admixture of coal dust with foreign material. These tests indicated that the presence of ash up to 40 per cent. of the dust of the kinds experimented with did not prevent propagation; that in one instance 50 per cent. did not prevent, but in another case did prevent propagation. In the later tests, at the experimental mine, it was found it required a larger percentage of ash to prevent ignition and propagation. See p. 584.

Concerning the bearing of the carbon-hydrogen ratio in ignition and propagation, systematic tests like those conducted in France by Mr. Taffanel were not made on account of insufficient facilities, but those that were made at the station, and subsequently at the experimental mine, confirm his findings rather than those of the English investigators, that the ratio is of great importance; that is, with other things equal (ash, moisture, and size of dust) the higher the percentage of volatile *combustible* matter the more easy is it to obtain ignition, and also propagation, though this does not mean that the violence of an explosion once started is greater with a dust of high volatile content. It means that the flame of the source of ignition must be hotter or of longer duration with dusts with low volatile content.

A decrease in volatile matter, speaking broadly, decreases the range of explosibility, into which enter the factors of ash, moisture, size of dust particles, structure of particles. A coal dust in which the fixed carbon is more than four times the volatile combustible is harder to ignite by a blown-out shot, or an electric arc, than one in which the ratio is 2 to 1 or 3 to 1. Anthracite that has a ratio of 10 to 1 or over has so far not proved to propagate an explosion even with a small percentage of firedamp present, but semi-anthracite in which the ratio is less than 10 to 1 has shown propagation with 2 per cent. of firedamp present, but not without it.

Welsh anthracite dust is considered by the British investigators as explosive, and they cite experiments in which charcoal dust proved explosive, to show that there is danger from anthracite dust without regard to the volatile matter present.

It was found, as had been anticipated, that the gallery was too short, being only 100 ft. long, to carry out explosion experiments which could be considered at all conclusive as regards propagation. From the beginning it had been expected that large-scale experiments would be carried

on in the underground entries or headings of a mine as soon as one could be secured. Accordingly, the following year, 1910, plans were definitely made looking to securing such a mine. No old mine suitable for the purpose was found, and it was finally decided to open up a mine by stages, primarily for the purpose of coal-dust explosion tests. The mine was opened when the state of the allotments permitted, and was prepared for the preliminary test at the end of 1911.

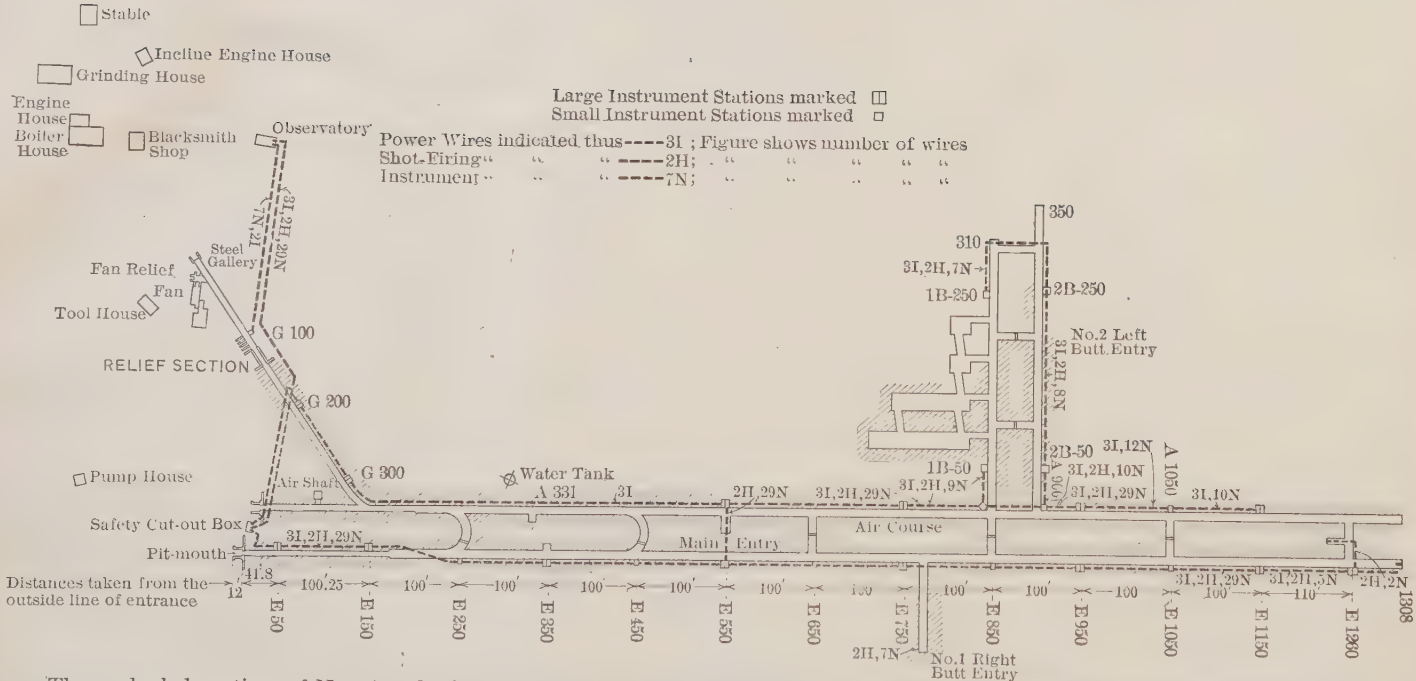
Fifteen explosion tests of a preliminary nature were conducted[3] in 1911 and 1912. The work was then suspended on account of the funds being exhausted. The mine was enlarged the early part of the fiscal year 1912–13 and a few experiments conducted during the sessions of the International Mine Experiment Station Conference, held at Pittsburgh, in 1912. For the fiscal year 1913–14 funds were allotted which permitted carrying on a most important series of over 100 mine-explosion experiments, the results of which are now being prepared for publication.

The mine consists of two parallel entries (see Fig. 1), driven into the Pittsburgh coal bed from the outcrop. These are now 1,300 ft. in length. The right entry of the pair is called the "main entry" and the left entry is called the "air course." There is a stub 100 ft. in length, turned at right angles off the main entry at a point 775 ft. from the mouth; and a pair of butt entries turned at right angles off the air course, the first of which is 850 ft. from the mouth. During the recent tests, termed the "second series of tests," these entries were 200 and 250 ft. respectively in length, but each of them has been extended 100 ft. during this past summer. Two rooms had been turned at right angles off the first butt entry during the second series of tests, and now two more rooms have been turned.

At 150 ft. from the mouth of the air course a diagonal entry comes in from the outcrop, and at the outer end of same there is a 120-ft. steel gallery, which by several offsets is connected to the fan. The outer 200 ft. of the main entry, air course, and slant are lined with reinforced concrete, but the early explosion experiments badly ruptured and broke up this lining and lifted the roof over the entries for 150 ft. from the respective mouths. The inner parts of the entries are like ordinary entries in the Pittsburgh bed, 9 ft. wide and from 6 to 7½ ft. high. The coal bed itself varies from 5 to 5½ ft. in thickness, over which there is a draw slate from a few inches to 1½ ft. thick, and above this a top coal, which so far has furnished a very good roof, although somewhat shaken in places. Fortunately, so far, very little timbering has been required.

In the main entry, and in the inner part of the air course, large instrument stations have been made every 200 ft. and intermediate to these small flame and pressure circuit-breaker stations. The main instrument

[3] See *Bulletin No. 56, U. S. Bureau of Mines*, First Series of Tests at Experimental Mine.



The unshaded portions of Nos. 1 and 2 Butt Entries show extensions since the second series of tests were concluded in July, 1914.

FIG. 1.—DIAGRAM OF EXPERIMENTAL MINE, SHOWING OBSERVATORY, STATIONS, AND CABLES.

stations are chambers constructed in the rib and lined with concrete, with heavy steel plates shutting off the entry, through which the instruments are connected with the mine atmosphere. Doors are placed to one side, permitting entrance in behind the instruments, which are placed upon shelves. The chief instruments are the pressure manometers electrically rotated for recording the varying pressures, as the explosion passes. Pressure circuit breakers and flame circuit breakers for interrupting electric circuits are placed in the main instrument stations, and also in the small intermediate stations. The stations are connected by wires formed into cables, which pass through pipes placed with compressed air and water pipes in a horizontal groove in the rib. This is faced up with concrete for protection of the cable and pipes. The wires go to the observatory on the surface, where the recording instruments are located. It is from this observatory that the explosions are started by firing through an electric circuit blown-out shots of black powder from a cannon. Premature firing is safeguarded by cut-out switches at the mouth of the mine and at the observatory.

The mine is equipped with six pressure manometers, three being like those used in the British experiments, and three like those employed in the French tests. These may be moved from station to station, in accordance with how the tests are arranged. In the British manometers, the pressure is transmitted through a bath of oil to a plunger operating against a triangular spring which carries the scribing style and that traces the pressure curve on a band of smoked paper, wrapped around a revolving cylinder. In the French manometers, the pressure acts directly on a small plunger which presses on a small deflecting mirror. A beam of light from an incandescent lamp within the apparatus is reflected on to a photographic paper around a revolving drum. The movement of plunger and mirror is so small that the inertia is negligible. For both kinds of manometers there are electrical devices and timing arrangements for marking the time on the pressure records with reference to the igniting shot.

Some modification, of these instruments have been made, and it is expected that others, embracing new features, will be constructed at the Bureau's instrument shop.

Connected with the mine there is a grinding plant for making coal dust, very large amounts of which must be used in some tests. This also serves for grinding shale dust or rock dust for mixing with the coal dust in determining the explosiveness of mixtures of different percentages of ash and for use in explosion-checking devices.

The fan is relatively large for a mine of this size. It will either force or exhaust about 50,000 cu. ft. of air per minute, and is quickly reversible. The offsets in the air passages, with a number of explosion doors, permit running the fan either way desired while an explosion is taking place.

A natural-gas pipe line passes near the mine, and a connection has been made to same so as to permit the introduction of gas into the mine.

The mine itself is in what might be termed a "peninsula" of the coal bed, being surrounded by the coal outcrop, except on one side parallel with the main entry. Firedamp is not found to an appreciable extent. Many analyses of samples taken even when there had been no ventilation for some time have not shown more than a few hundredths of 1 per cent. methane and, with a moving current, none, or occasionally a trace, can be detected by analysis.

Method of Carrying on Coal-Dust Explosion Tests

Loading of Coal Dust.—The coal dust for the tests cannot be placed on the floor of the entry, as is the practice in European galleries, because the floor is frequently wet, and, as is the case in commercial mines, the track road bed is more or less rough. In all the tests except recently the coal dust was placed on side shelves of 3 by 4 timbers, fastened by bolts to posts recessed into the ribs, or, in the concreted section, bolted directly to the concrete. There are five shelves on each side of the passageway. It was thought that the shelves being parallel with the path of the explosion, and being strongly bolted to the posts, which in turn are fastened by heavy iron bands into the rib, would not be damaged by the explosions. This, however, has not proved to be the case when the explosion is of a violent nature. This method of loading on the side shelves proved entirely successful when pure coal dust was used, and the mine was dry; but when the walls were wet, also in mixed-dust tests, all of the dust was not brought into suspension, and inconsistencies were found. Accordingly a system of cross shelves, placed close to the roof, was employed to supplement the side shelves. Such cross shelves correspond to the collars of timber sets in a commercial mine. It was found that this made a great change in the results of tests where the dust was near the border line of explosibility, and enabled the obtaining of consistent results, regardless of whether the walls and floor were wet.

Method of Ignition.—Explosions are initiated by blown-out shots, as is the usual method abroad. It is much the easiest method, as compared with employing a body of firedamp. In the earlier demonstrations the shots for blowing out were placed in the face and cased with pipe in order to insure the blowing out. This method, while successful, takes more time to arrange, and also, as some energy is expended on the walls of the drill hole, the pipe usually being burst, more consistent results were obtained by the use of a cannon with a bore hole of about $2\frac{3}{4}$ in., and 20 in. deep. Black powder was used in almost all cases, the charge usually being 4 lb., stemmed with clay. Circuit-breaker wires are placed across the mouth of the drill hole, so that the moment of firing may be recorded. As

already stated, the shots are fired electrically from the observatory, and switches at the mouth of the mine cut out all wires entering the mine until every one has left the mine. The mining engineer in charge and the mine foreman connect up the shot-firing wires to the detonator leads after every one else is out and then come out themselves.

Point of Ignition.—In the majority of tests the ignition shot was placed at the face of the main entry, pointing outward, parallel to the axis, the coal-dust loading, either pure or mixed, extending outbye from the shot in whatever arrangement may have been planned. The cannon is placed either on a mine truck, or latterly on the floor of the gallery, and a platform of boards, nailed to the ties, extends outward from the cannon. On this, 25 lb. of dust is scattered.

In a few cases two cannons were used, and a cross fire obtained, but the shock waves from the two shots complicated some of the manometric records so that a single shot was considered preferable. In some of the later tests the ignition point was in a short stub off the last crosscut midway in the pillar between the two entries. This was in order to secure double tests, using both the air course and entry. These entries furnish symmetrical conditions. In two tests the ignition shot was located at the head of the first left stub, and in a few other cases there was special placing of the ignition shot.

Necessarily many tests were made to standardize the method of ignition, the method of loading, and the instruments. The first objects were to find out how coal dust was ignited, and how the explosion was propagated from point to point; the pressure waves set up by the explosion; the movements of the pressure waves; the relation of the rise and fall of pressure at different points in the course of an explosion; the position of the flame with reference to the pressure waves; the velocity with which the pressure waves and the flame travel under different conditions, and the character of the combustion as far as that could be determined by the gathering of gas samples automatically at different stages of the explosion.

The second object was the determination of the effect of mixtures of shale dust or limestone dust on the ignition and upon the propagation. In this connection, the term "ignition" means the prolongation 50 to 100 ft. or more of flame originated by the shot; the term "propagation" refers to continuation of flame throughout the mixture or dust being tried.

At this point it may be well to explain, as far as our present knowledge goes, the way in which an explosion starts and propagates. When the igniting shot of black powder is fired, a flame is projected from the hole for 10 to 15 ft. the distance depending on the size of the charge and size of the grains of powder. This starts a shock wave, which travels at the rate of a sound wave. Tests in the mine have determined that this travels in the passageways at an average speed of 1,150 ft. per second. The effect on the sensitive French manometer is to cause a sharp rise

from the zero line to a point where a pressure of 3 to 4 lb. is indicated. This line is jagged, showing vibratory movements of the wave. The shock-wave pressure decreases in intensity, though not in velocity, as it progresses, and is barely discernible on the pressure instruments (which measure the side pressure only) located 500 to 600 ft. from the cannon. A portion of the dust on the floor platform is raised by the blast, and is ignited by the flame of the explosive. This burns rapidly, and other dust brought from the side shelves by the eddying of the air currents is carried into the zone of combustion. As far as the manometers show, there is an interval of $\frac{1}{2}$ sec., more or less, before the pressure produced by the burning dust becomes appreciable. If the zone of dust loading is short, or if there is a high proportion of ash so that the mixture is barely explosive, the initial pressure near the origin will not be more than a pound or two, but with pure dry dust the pressure will rise rapidly. The flame from the burning dust takes from half a second to a second, depending on the conditions, to traverse the first 100 ft. In a pure coal-dust explosion, the second hundred feet is traversed more rapidly, that is from $\frac{1}{2}$ to $\frac{1}{4}$ sec. or less, and from this point onward the velocity rapidly increases until velocities of 2,000 ft. per second and upward are attained. As the explosion progresses in an explosive dust cloud, the pressure rapidly rises, and at a point 350 ft. from the origin pressures of 63 lb. per square inch have been attained, at 550 ft. from the origin pressures of 73 lb., and at 750 ft. 119 lb. per square inch have been reached. Where an explosion is abruptly checked by means of barriers it is found that the maximum pressure is attained at the moment of passage of the flame, but there is a pressure wave in advance of the flame. This is the unfortunate feature which permits propagation of a dust explosion, inasmuch as the dust is raised by this pressure wave. The English term it the "pioneering wave."

When the coal dust is uniformly loaded, and is of a highly explosive character, the pressure from point to point keeps on rising, as far as can be determined within the length of the passage of the experimental mine; as this increasing pressure wave advances, reflex waves are thrown off which go back toward the origin through the column of moving gases. In other words, at the area of high pressure, waves go in both directions to the areas of low pressure. This explains many of the anomalous features which have puzzled investigators of disastrous mine explosions. That is, evidence of violent movements in the direction opposite to that of the explosion are sometimes observed. At the experimental mine large wooden blocks, marked for identification, have been placed at regular intervals, and in an explosion of increasing intensity it is found that up to the point where the maximum pressure has been reached the blocks have gone toward the origin, and beyond this point, outbye or in the direction of the advance of the explosion.

The same phenomenon has been demonstrated in some tests by placing loaded mine cars at intervals along the passageway; these have been moved toward the origin in the instances where the maximum pressure was attained beyond, although doubtless they first moved outward as the explosion wave passed before being hurled inward. In one instance a loaded car was thrown toward the origin 147 ft. and overturned.

It is frequently a matter of speculation in the investigation of explosion disasters, what pressures may be attained to produce the disruptive effects observed. With abundant inflammable dust there seems to be no limit, as the effect is cumulative. Let us suppose the pressure has attained 120 lb. per square inch, or about eight atmospheres, there are then eight times as many atoms of oxygen present in a cubic foot as there were in a cubic foot at the initiation of the explosion, so that if there is enough dust present to combine with all the oxygen there is eight times the energy liberated by the combustion per unit of space as at the beginning of the explosion.

As the explosion attains high velocity, the compression of each successive layer of air ahead is probably nearly adiabatic, so that the temperature of the air and dust is instantly raised to the point of combustion. Therefore actual transfer of heat from point to point by radiation and convection, which would be necessary at the beginning, might not be when the velocity of the explosion was 2,000 to 3,000 ft. or more per second.

Composition of Afterdamp

The investigation of explosion phenomena would not be complete without obtaining samples of the resultant gases and analyzing and correlating them. In the French gallery the samples were taken in evacuated flasks, the glass stems of which projected into the gallery and were ruptured by the explosion and the gases forced into the flask. Observers could immediately after the explosion turn off the stop cocks of the flasks. This method was not practicable in the mine, so various arrangements were made to seal automatically the flask connections, but there was some uncertainty about the time the sealing was done.

In the British Altofts tests automatic sampling bottles were employed and a similar one was tried at the experimental mine, but it did not prove suitable to the conditions.

A special automatic sampler was then devised and built, in which there are three evacuated steel bottles, the glass ones first used being frequently broken, although put in a recess in the rib and protected by a heavy steel cover. Each bottle has the end closed by a thin plate of glass; a special mechanically operated plunger breaks the glass and seats itself like a cork in the metal neck. This mechanical arrangement is set in motion by the interruption of an electric circuit through the burn-

ing out of a tin-foil strip exposed to the flame of the explosion. A clock work controls the time of operating the mechanism of the second and third bottle, which can be regulated to obtain any d sired time intervals. The time between the burning out of the tin-foil strip and the closing of the first bottle is very small; tests have determined it to be but 0.03 sec. It could not be made less and obtain a full sample in spite of the evacuation of the bottles and the exterior pressure of the gases; as it now operates sometimes the bottle is only partly filled at atmospheric pressures.

At first the tin foil for starting the operation was placed at the sampling point, but in recent tests it has been placed 100 ft. nearer the origin in the effort to secure samples either in the first or second bottle, which would show the preliminary stages of the combustion. The difficulty of getting decisive results regarding this early chemical change is that the samples necessarily are taken on the side of the passageway. The desired point would be at the center, but the violence of the explosions makes this difficult, though some method may yet be devised of accomplishing it. It is known from Mr. Taffanel's investigation in his surface gallery that the flame of the explosion first advances on the center of the passageway, the point of flame being from a few feet to many feet in advance of the flame at the walls.

The analyses of the samples at the experimental mine are most interesting, but it is too early to draw conclusions therefrom. Two typical sets of analyses from the automatic sampler are as follows:

Test No.	No. of Sample	Seconds[b] after Passage Flame	CO_2	CO	O_2	N	CH_4	H	C_2H_4
92	1	0.04[c]	13.56	5.36	1.03	75.21	1.15	2.38	1.31
	2	0.55	7.93	5.14	7.05	76.34	0.97	2.45	0.12
	3	3.00	9.98	7.38	2.60	74.76	1.51	3.52	0.25
114	1[a]	+0.04[d]	0.60	0.35	19.82	79.18	0.05	0	0
	2	0.33	0.71	0.20	19.59	79.20	0.30	0	0
	3	1.53	7.55	6.92	4.35	75.96	1.41	3.63	0.18

[a] Normal air.

[b] Time of closing valve after circuit broken by flame.

[c] Probably sample gathered later through lag of valve.

[d] In advance.

Perhaps the most interesting feature of many of the analyses taken during the series of tests is the increase in carbon monoxide in the second or third sample over that in the first or second sample gathered in the respective test.

Another interesting feature is the distillation of the coal dust after the first passage of the flame. In Test 114, the analyses from which are given above, the first sample was taken before the explosion reached the sampler, the tin-foil circuit breaker having been advanced 100 ft.; the second sample seems to have been taken immediately following the first dart of flame before the combustion was general. The velocity of the flame for the 100 ft. preceding was 1,408 ft. per second as registered by the flame circuit breakers.

Development of Mine-Explosion Prevention Methods

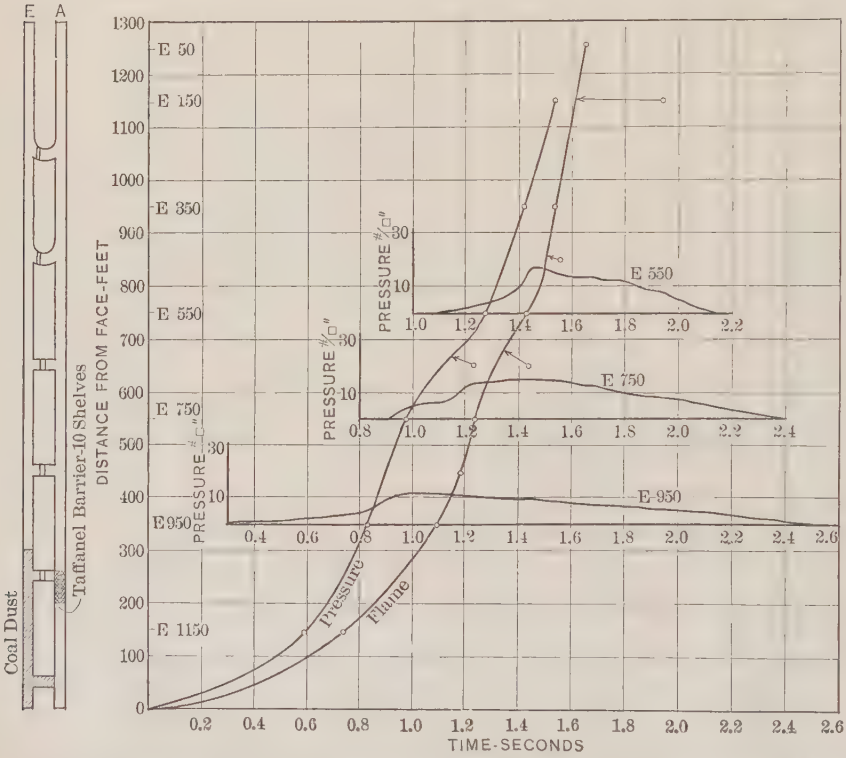
It is needless to say that this extensive and costly experimentation would not have been undertaken for mere demonstrations that coal dust is an explosive agency (necessary as the educational work was for the first year or two). The final object was to obtain such knowledge of the phenomenon that practical methods could be devised for preventing or limiting mine explosions. The first consideration of operators, naturally, is the prevention of ignition of coal dust by use of the well-known though often neglected precautions already available, such as the employment of safety lamps in mines making gas, the use of permissible explosives, safe-guarding electrical installations, lessening the production of coal dust, rigid inspection, etc. These features will not be gone into, but we will confine the present discussion to the methods of making coal dust inert, for in bituminous and lignite mines the dust is always with us, so it must be made harmless in one way or another; and to explosion checking or limiting devices.

Tests in the Pittsburgh gallery, and many practical illustrations in coal mines of the United States, had already demonstrated that coal dust so damp that it cannot be brought up into the air by concussion, is not dangerous. The way to judge whether this condition is attained, suggested by the writer of this paper as a result of early Pittsburgh tests, is to gather a handful of the dust: if it will stay balled up after packing in the hand it will not be raised in a cloud by a concussion or blast, and therefore will be harmless.

Methods of wetting coal dust have been experimented with at the experimental mine, and, as in commercial mines, great difficulty was found in wetting pure coal dust when it lay in piles, either by humidifying or direct watering. As every coal-mining man knows, coal dust repels water, and water may run over it or under it, like quicksilver, without wetting the dust, even after months of contact. However, in commercial mines where the walls and floor are thoroughly wet, the particles of dust falling on the road will tend to become damp. One fact has recently been found, which seems to be of great value in aiding the wetting of dust by watering or humidifying methods, namely, that where coal dust

is mixed with shale dust or limestone dust, the repulsion of the coal dust is nullified and water quickly penetrates masses of the mixed dust. This suggests that where watering or humidifying methods are used, it would be of advantage to scatter fine shale dust through the mine, or preferably limestone dust on account of its whitening the ribs, roof, and floor, thus making the illumination of the passageway easier.

Dustless conditions have been proposed as a preventive of dust explosions, to be obtained by attempts to clean carefully the surfaces, and



Coal-dust loading 300 ft.; outbye this the entry for 1,000 ft. to the mouth of mine was cleaned by sweeping and use of compressed-air jets. The explosion traversed the so-called dustless zone, and flame issued from the mouth of the mine. In the air course, 150 ft. from the last open cross cut there was a Taffanel barrier, which stopped the explosion in that direction.

FIG. 2.—DIAGRAM OF TEST 79.

possibly through the employment of vacuum cleaners. The latter have been tried in Europe, but so far have not proved effective, as there was great difficulty in cleaning up coal dust from the rough surfaces of roof, walls, and floor. Striking illustrations of the ineffectiveness of the most careful cleaning have been demonstrated by the attempts to secure dustless zones for testing purposes at the experimental mine. It is neces-

sary to clean very carefully between experiments, and this is done by means of shovels, brooms, blowing of dust from the walls and floor with compressed air, aided by the ventilating current. In spite of these precautions, in tests with ignition zones of coal dust at the head of the main entry of only 100 to 300 ft. in length, upon being ignited, the explosion has gone 500 to 1,000 ft. through carefully cleaned entry when the walls of the entry have been dry. In one such instances the explosion increased in intensity as it went out (see Fig. 2). There is another factor which enters into this besides the difficulty of cleaning the fine dust out of crevices and recesses in the ribs, namely, that the air wave in advance of the flame carries timbers, pieces of rock, coal and other débris, which abrades the ribs, and it seems possible, from the tests at the experimental mine and observations made in mine disasters, that this may be an additional source of fuel for the explosions, especially in mines where the coal is friable. In considering this it must be remembered how little dust is necessary to propagate an explosion.

As already mentioned, on one occasion 0.032 oz. of coal dust in a cubic foot of space was ignited in the Pittsburgh gallery. Mr. Taffanel reports one ignition with 0.023 oz.; and obtained explosions quite regularly with most dusts he tested, employing only 0.070 oz. per cubic foot of space. The cross-section of the experimental mine contains about 60 sq. ft., which for the amount of 0.070 oz. per cubic foot of space would require only 4.2 oz. per linear foot of entry. This amount distributed on the floor and the two ribs would make but 0.19 oz. per square foot of surface, or a layer of dust less than 0.003 in. in thickness. Most of the experimental mine tests are made with five times this amount of dust in order to insure maximum conditions for an explosion.

Many persons do not realize how thick a dust cloud has to be to obtain ignition which propagates beyond the immediate influence of the igniting flame. It has been found in laboratory and surface gallery tests at Pittsburgh, that a cloud of dust which will ignite appears very dense to the eye; in fact, in small-scale tests made with electrical ignition it is found the cloud must be so dense that it entirely obscures a 16-c.p. light placed a foot from the glass window. Coal dust is capable of extremely fine subdivision. It has been shown by laboratory tests, as well as gallery tests, that the finer the dust is ground, the more readily it is ignited. Dust through 200 mesh is the standard for testing in the laboratory inflammability apparatus at the Pittsburgh station. In the coal dust used in the experimental-mine tests, about 80 per cent. of it will pass through a 200-mesh sieve. The openings of a standard 200-mesh screen are 0.0029 in. square. If it be assumed that the average size of the particles of dust passing through the 200-mesh sieve is equivalent to a sphere of one-half the diameter of the screen opening, or, 0.00145 in., in a cubic foot of such dust cloud weighing 0.07 oz. there

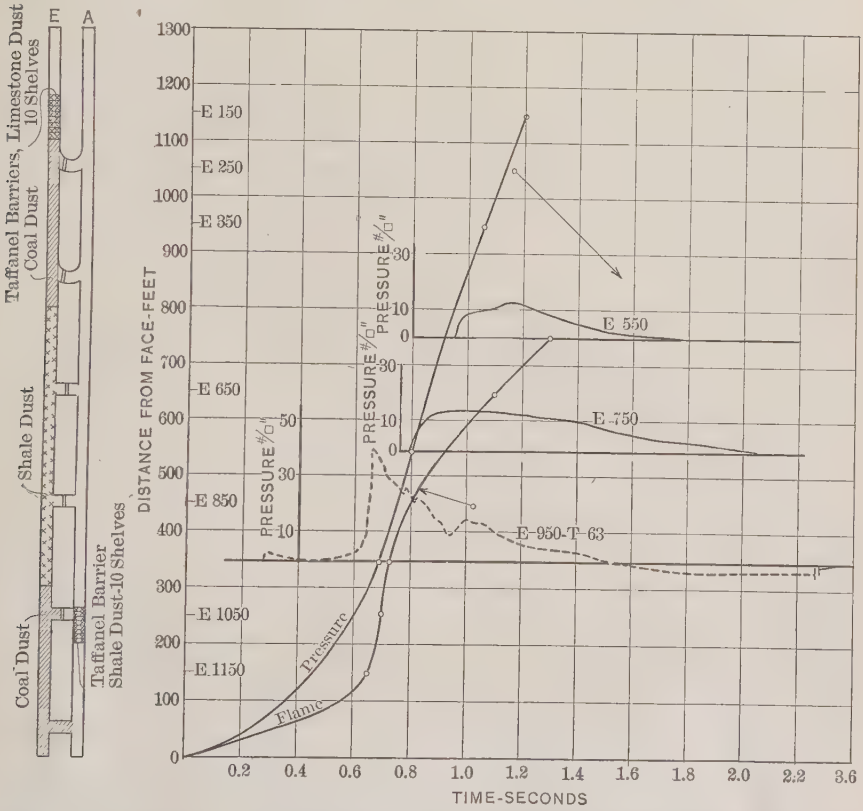
would be 2,735,000,000 such particles. Microscopic study makes it seem probable that there is a much larger number of coal particles in a total weight of 0.07 oz. The vast number of dust particles that may exist in such a dust cloud makes it easier to comprehend that a cloud of fine dust behaves essentially like an inflammable gas.

Dry dustless zones, at least where these are in entries having coal ribs, proving futile to stop or even check an explosion; the alternatives are to wet the dust, or to use inert dust. Watering throughout a mine, if thoroughly done, is unquestionably a preventive of coal-dust explosions. Short watered zones, however, are not effectual, as the excess of dust from the zone behind is borne along by the advance wave. The explosion tests at the experimental mine which have been made during the spring and summer months, through natural agencies, have had to be conducted with thoroughly wet walls and floor, the side walls and roof being so wet as to be slimy to the touch. When the coal dust has been placed on shelves, particularly cross shelves, the wetness of the walls and floor has not had an appreciable effect in checking the violence of the explosion. Further, the flame has frequently penetrated along dampened zones 500 to 600 ft. in length, there being sufficient coal dust carried along by the explosion to sustain combustion throughout such zones. Many similar instances are found in explosion disasters, which have unfortunately led some to believe that all watering is ineffectual; rather, it points to the necessity of having the watering extend throughout the mine.

Rock-Dust Treatment.—The use of rock dust as a preventive of explosions was first urged by William E. Garforth, as a result of his observations following the Altofts explosion disaster of 1886, that certain airways covered with rock dust, which naturally drops from the roof and sides in this long-wall mine, were in no instances penetrated by flame. It is generally claimed that in British coal mines, also in French and Belgian mines, extensive watering will seriously damage the walls and roof; although in Germany thorough watering is compulsory, and sprinkling is more or less used in South Wales.

Largely as a result of Mr. Garforth's influence the Altofts gallery was first established, and one of the objectives was to try out rock-dust treatment. Resulting from the favorable tests, the method was adopted immediately at the Altofts colliery, and subsequently at a number of other collieries in Great Britain. The method was therefore tried at the experimental mine, largely in view of the fact that many mine operators of the United States claimed that watering methods are inefficient, and investigation has found that only a very small fraction of the mines, even where watering is done, wet down the ribs and roof, where much of the most dangerous dust (the float dust) collects. Again, in the arid Rocky Mountain regions, the great scarcity of water and extreme dryness of the

air entering the mine make watering a particularly difficult problem. Several great explosion diasters, which have occurred in that district in mines where considerable watering has been done, indicate the need of providing other means than watering in mines in which there is great difficulty in properly wetting the dust. In fact, one of the greatest disadvantages of watering methods is that so much depends upon the individual men employed to do the watering, for it must be done, not occasionally, but every day in the year.



Pressure at station 950, 40 lb. Flame penetrated through the shale-dust zone 500 ft., but did not ignite the coal-dust loading in the zone beyond.

FIG. 3.—DIAGRAM OF TEST NO. 86, TO DETERMINE PENETRATION OF SHALE-DUST ZONE WITH A STRONG INITIAL EXPLOSION OF PURE DUST.

Humidifying by steam jets is not open to this latter objection, and this method has been found to be excellent in wetting the roof, ribs, and floor in tests at the experimental mine, although, as already stated, the pure dust does not become wet when it is in undisturbed layers or piles.

In the rock-dust tests at the experimental mine shale dust, obtained by grinding roof material, was chiefly employed. This was not ideal in

that it contained from 8 to 10 per cent. of combustible matter. It would be better if shale could be obtained without combustible matter, as is the case with many of the roof shales found in Great Britain. However, as it is a material which is convenient and available throughout the fields mining the Pittsburgh bed, from which a large percentage of the bituminous output of the country is derived, it was thought best to test it out thoroughly, especially since if effectual the proportions of shale to coal dust would be on the safe side. This shale dust, distributed along side shelves, was successful in stopping small initial explosions of coal dust in a few hundred feet; but with an initiating zone of 300 ft. of coal dust, with resultant high pressures of 30 to 50 lb. per square inch on entering the shale-dust zone, the flame penetrated from 250 to 650 ft.; as, for example, in mine test No. 86 (see Fig. 3 showing flame extent and velocity, also pressure records at certain stations), in which the initial pressure from the coal dust was 40 lb., the flame traveled for about 300 ft. before dying away. In Test 87, in which the pressure was but 30 lb., it died away in less than 250 ft.

Limestone-dust screenings, which are a waste product in many districts, were also tried in zonal treatment, but owing to the greater specific gravity of the limestone, the results were no more effectual than with the shale dust placed on the side shelves, although the limestone contains no combustible matter.

The conclusions drawn from these tests, and others in which the rock dust was placed on cross shelves, are, that the most effectual method is to place the dust on cross shelves, and on overhead timbering, if there be such; then the advance air wave causes the entire dust load to fall in a shower and practically all of it is brought into play, whereas when placed along the ribs and floor it is not so easily brought into suspension when the explosion pressures are low. With this cross-shelf arrangement, which is practically a continuous Taffanel barrier, the greater specific gravity of the limestone dust has no appreciable disadvantage since, after tests, it is found widely scattered, and on the other hand there is some liberation of carbon dioxide, which further tends to quench the combustion.

In connection with the extinguishment of the flame of an explosion, it is interesting to note that in an effort to start a strong ignition of coal dust 75 lb. of coal dust was placed on eight cross shelves from immediately in front of the cannon to 50 ft. distant. To the surprise of all present at the test, the flame did not pass beyond these cross shelves. The test was repeated five times in all, with the same results except in one case when propagation occurred. The representatives of the foreign testing stations were asked if they had experienced such results. Subsequently M. Watteyne of Belgium reported he had tried the same experiment, and obtained confirmatory results. Coal dust placed in this way appeared to have acted like a rock-dust barrier; the coal dust falling rapidly across

the path of the flame cooled it below the ignition point of the dust. It must be clearly understood, however, that an excess of coal dust is not a protection against an ignition under any ordinary circumstances, as it is almost impossible to conceive of its being similarly placed under working conditions. If there is merely an excess of dust on the floor, or in the gob, only a portion of it rises in suspension, which would ignite and propagate if other conditions were favorable. In other words, it was not simply an excess of dust, as there was no more than in the usual test, but it was the way it was placed so that it dropped across the initiatory flame as a mass, and absorbed the heat. It was interesting to note that in each case there was much burning dust found on the platform in front of the cannon after the test. When there is an excess of coal dust along the entry this is not a hindrance to an explosion, as the advance waves carry it along until the flame reaches the dust; they automatically drop it if the waves slow up. Theoretically, 0.123 oz. of Pittsburgh dust, if burned completely, will combine with the oxygen in a cubic foot of space at ordinary pressures. However, when an explosion becomes violent as it advances along a passageway, the pressures may rise to 7 or 8 atmospheres or more; the dust cloud at such a point must contain a proportionately larger amount of dust to use up or combine with all the oxygen in a given space. It has been observed that in most violent mine explosions there is usually a great excess of dust over that necessary to combine with all or most all of the oxygen in the zone of violence.

Rock-Dust Barriers

Despite the greatest precautions taken, it is inevitable that dust explosions will occur from one cause or another, and on this account it is deemed advisable that there should be a secondary line of defense to limit or stop them from going out of the originating district. Mr. Taffanel, director of the French Station, was the first to plan what has been termed a "barrier." He tried heaps of dirt across the passageway and observed that they had an important effect in choking explosions Manifestly such methods could not be used in mines, so he devoted himself to making arrangements which would give equivalent results. This led to the adoption of the system of cross shelves placed close together, about 2 yd. apart, on which shale dust, flue dust, or fine ashes was placed, each shelf being $\frac{1}{2}$ m. (20 in.) wide. At first he specified 10 such shelves, but later as a result of further experiments he found it advisable to increase the number to 15. The principle of this is that the air wave in advance of the flame launches the large mass of incombustible dust previously placed on the shelves, into the air and makes a dense cloud, which cools the flame of the explosion. This arrangement was adopted by many of the mines in France. At the experimental mine the barriers, which in com-

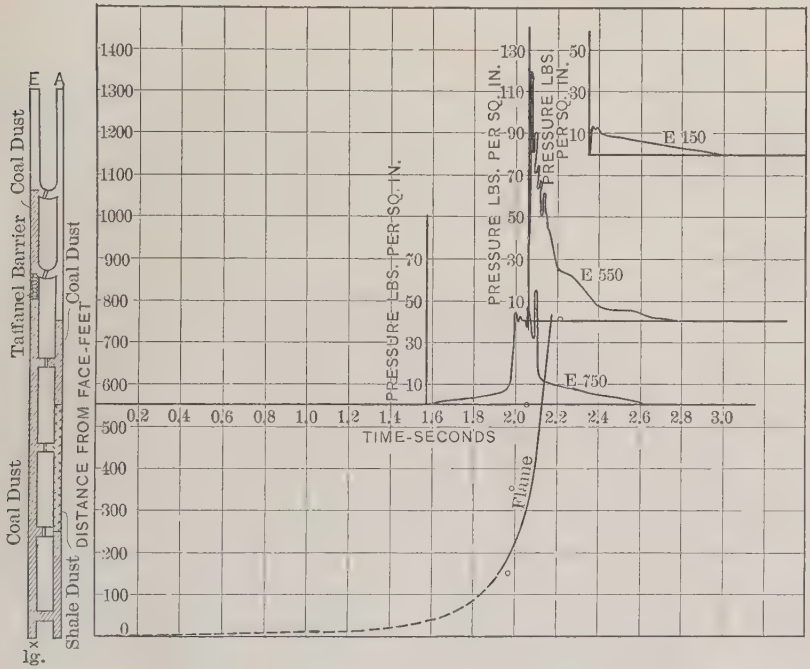
pliment to Mr. Taffanel are termed "Taffanel barriers" (Fig. 4), were employed to limit the zone of violence, as otherwise it would have been impossible to make tests frequently, owing to the extensive damage which results from explosions of pure coal dust, when permitted to traverse over 500 or 600 ft., to the interior of the mine and the roof and concrete linings near the entrances; the arched linings have been badly disrupted by some of the extensive explosion experiments. In Fig. 5 is shown a diagram of a

FIG. 4.—TAFFANEL ROCK-DUST SHELF BARRIER.

test in which the flame of the explosion was extinguished by the rock-dust barrier, the pressure dropping from 119 to 12 lb. in 400 ft. The barriers proved uniformly successful in stopping explosions, except in a few instances of very light explosions or other inflammations where the flame passed the barrier. There is also the disadvantage that where the air current is strong, as it generally is along the haulage ways of American mines, the dust is blown from the shelves. Again, in mines where the watering system is employed, the dust will likely become dampened, and when wet it is of course ineffectual in quenching explosions.

These disadvantages led the writer to design some special forms of inclosed barriers which would operate with lighter pressures than the Taffanel open barriers. There are two forms of these: One the box barrier, where six boxes each containing about 700 lb. of dust are hung

from the roof and timbers across the entry. In the final design (Figs. 6 and 7) the arrangement comprises a light open box hung by four freely suspended rods, with eyes, from open hooks fastened in the roof or overhead timbering. The box within limits may swing freely; when, however, the movement is over $\frac{5}{8}$ in. or thereabouts from the middle position, the two suspending rods on the side toward which the movement is directed, which fasten to the bottom edge



Explosion test of pure Pittsburgh coal dust, 800 ft. zone, ignited at face of main entry. Explosion started with unusual slowness, but developed into one of great violence, maximum pressure at station 550 being 119 lb. The explosion in main entry was stopped by the Taffanel barrier at end of coal-dust zone. The coal-dust zone outbye this was not ignited, and the pressure fell from 119 lb. at station 550 ft. from mouth to 12 lb. at station 150. In the air course the coal-dust loading extended for 250 ft.; outbye this there was a 300-ft. zone of shale dust, with coal dust beyond. The flame penetrated through the shale-dust zone, but did not ignite the coal dust beyond. The explosion showed very violent results in the interior of the mine.

FIG. 5.—DIAGRAM OF TEST 46.

of the box, are struck by cleats fastened to the upper edge of the box, and if the movement is sustained by the advance wave of the explosion, the eyes of the suspending rods are pushed off the open hooks and the box swings downward, pivoting on the points of suspension of the rods on the opposite side of the box. The rock dust in the box is partly discharged, but some is held by the bottom of the box, which is constructed of two loose boards resting on cleats attached to the side of the box. The boards are hung

by chains attached to the roof or overhead timbers, which permit the boards to drop 3 or 4 in., then they are caught by the chains so that a portion of the rock dust is held on them, unless the explosion is so violent that all parts of the box are ruptured. In the latter event the rock dust is automatically launched and mixed with the air and coal-dust cloud.

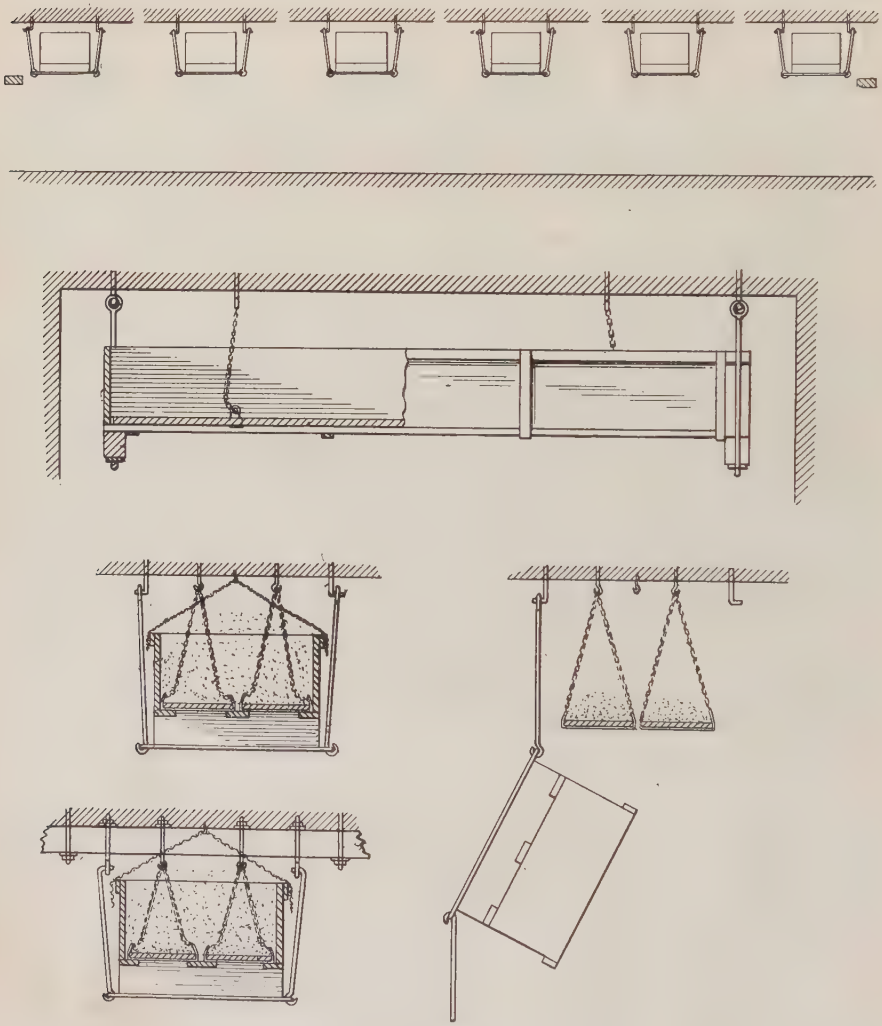


FIG. 6.—RICE BOX BARRIER.

The arrangement of hanging the bottom boards, which allows some of the retained dust to sift down slowly, is to take care of the more difficult situation brought about in light explosions, when there may be a considerable time interval between the advance air wave and the flame. It

has the further merit that if a box is accidentally tripped off, the bulk of the weight is caught by the chains, and it is not believed a serious injury would be caused to a man who might be caught underneath.

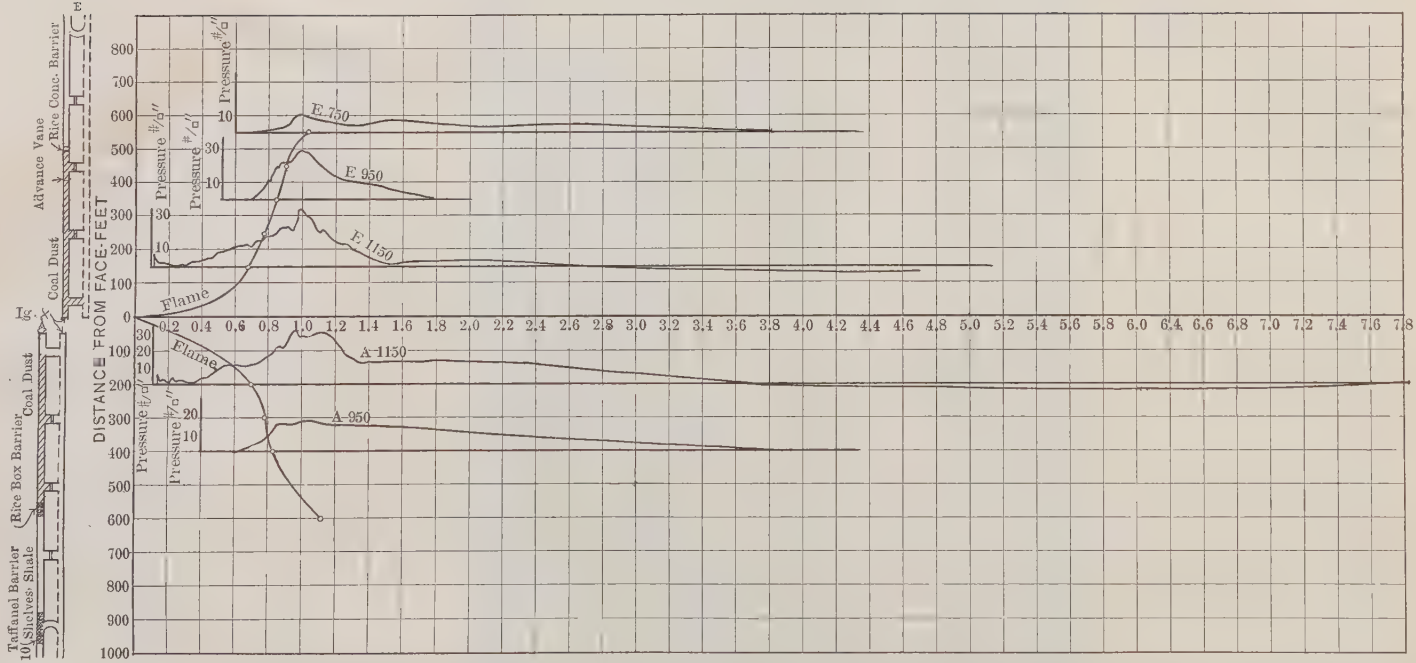
To protect the rock dust, in an ordinary mine working, from becoming damp or admixed with float coal dust, the box should be covered by a waterproof cloth attached over the center line of the box to the roof, so that as the box dumps the cloth is automatically pulled from its top.

As indicated above, there is less difficulty in stopping a violent explosion than a light one, because in the violent explosion the whole

FIG. 7.—BOX ROCK-DUST BARRIER.

mass is disrupted, and the rock dust forms a thick cloud of inert dust. Many tests have shown that the rock dust from the boxes is very widely distributed; and when limestone dust is used it can be observed ejected from the mouth of the mine by the air blast.

Another form of barrier, which is illustrated by Figs. 9, 10, and 11, is termed the "concentrated barrier." In this the rock dust is placed on a large platform hung close to the roof. The platform consists of boards, hinged separately to a cross timber, and held up at the other end by an angle iron. This angle iron, in turn, is held up by a system of levers which are tripped by vanes. There are advance vanes connected by a wire with the lever which pushes upon the tripping lever. In the experi-



This shows a symmetrical explosion of pure coal dust in the main entry and air course to determine if the concentrated barrier and box barrier in the respective entries would stop the explosion. The explosion was stopped by them.

FIG. 8.—DIAGRAM OF TEST 105.

ments at Bruceton the advance vane has been placed about 100 ft. from the concentrated barrier. As erected in a commercial mine there would be a similar vane on the other side, as the barrier is symmetrical and will extinguish an explosion approaching from either side. In case of possible accidental tripping, and danger when a man or trip of cars is passing, the inner end of each of the platform boards is fastened by a loose chain to the roof, or to a cross timber, permitting only a limited drop. As hitherto arranged, half of the boards dropped about 9 in., and the other half about 18 in. The platform holds about 3 to 4 tons of rock dust, and is quite compact, occupying the space between timber sets, which are placed 7 ft. apart. In mines with low roof it may be

FIG. 9.—CONCENTRATED BARRIER, SHOWING UNDER SIDE OF SAME, AND BUMPING VANE IN FRONT; ALSO TRIPPING WIRE FROM FIRST VANE, 100 FT. IN ADVANCE.

necessary to brush down the top to make room. The appearance of the barrier in passing under it is like that of an ordinary overcast.

The leverage system of operating the concentrated barriers can be arranged so that it will operate with almost any desired pressure. As arranged at the experimental mine, it will not discharge until the pressure is equivalent to that of a wind of about 40 miles per hour (3,500 ft. per minute). In other words, a heavy gale. It is thought that the proper places for such concentrated barriers are at intervals in the main entries, and in each side entry near its mouth, so that if the mine is paneled, an explosion will be confined to one panel.

It is interesting to note that Mr. Taffanel had independently decided that it was desirable to have such inclosed barriers, and had designed a concentrated barrier just prior to his coming to Pittsburgh recently for collaboration on special tests. This was also tried out successfully at the experimental mine, but it did not have the advance vane for tripping it, and this Mr. Taffanel thought was a considerable gain to provide for very rapid explosions.

As far as the tests have gone, the box barriers and the concentrated barriers are equally effectual, and the adoption of one or the other at

FIG. 10.—CONCENTRATED BARRIER AFTER EXTINGUISHING A VIOLENT EXPLOSION.

the commercial mines will perhaps depend somewhat on local conditions.

The success of the foregoing barriers led to the designing of a rock-dust protected ventilating door, since ventilating doors are invariably destroyed in all explosions of any extent; and further, most of them being near the mouths of side entries, it would be of the greatest value to have an explosion wave arrested there (see Fig. 12).

The principle underlying the door is to have the inclosing frame work put together without nails, and keyed in position by the door frame, so that when the door frame is blown out the mass of dust held behind the side and top boards is dislodged, and forms a dust cloud.

Again, it was found advisable to sustain a certain amount of dust close to the roof, and this is accomplished by attaching to the boards over the

door frame loose chains, hung by bolts into the roof or to higher timbers. These sustain some of the dust after operation of the barrier and allow it to sift down in case there is a considerable time interval between the arrival of the pressure wave and the advance of the flame.

It has already been found that the door frame must not be made too light or it is likely to be blown down by the shock wave from the cannon. It can therefore be a substantial frame, well braced to resist the ordinary shock wave from blown-out shots.

FIG. 11.—CONCENTRATED BARRIER AFTER LIGHT EXPLOSION, WHICH IT EXTINGUISHED.

Another device with a similar object is a rock-dust protected overcast. As every one knows who has investigated explosion disasters, overcasts are particularly vulnerable, though built of concrete and iron. Even with a small pressure per square inch, the large surface exposed gives a tremendous total pressure, which it is practically impossible to withstand by any ordinary construction. Therefore, where natural overcasts cannot be made which will provide a good thickness of natural strata between the airway and entry, it seems just as well to provide comparatively light overcasts made of wood, or if preferable, of light concrete or brick construction, not reinforced, making the walls double, and filling same with rock dust, or flue dust, holes being provided for occasional inspection, and if necessary, renewal of the dust. By having

such rock-dust protection, if the overcast gives way by the pressure of an explosion, the flame is prevented from entering the airway. Oftentimes this might isolate a large district of the mine, and thus save the men in it.

To complete the devices for the protection of vulnerable points a rock-dust stopping was designed, which presents some unique features. The first tests were with hollow walls, but the objection to this was that the stoppings are likely to go out quickly at the first pressure wave, so if there is a considerable interval between this wave and the flame there may be a chance for the rock dust to settle and the flame to pass

FIG. 12.—ROCK-DUST PROTECTED DOOR WITH FRONT BOARD REMOVED TO SHOW LOOSE CHAINS TO SUPPORT BOARDS OVER DOOR FRAME IF EXPLOSION OCCURS. TAKEN BEFORE SUCCESSFUL OPERATION.

overhead; further, the stopping is liable to go out in a mass, and the rock dust would not be well distributed. In the design which was tested with successful results, and which is illustrated in Fig. 13, the ease with which stoppings are blown out and their vulnerability are taken into account. The dust protection is practically in two halves, one independent of the other, and advantage is taken of the angle of repose of the rock dust. Practically, the device in each half consists of a series of shelves one over the other, with a space between the two halves in which there may be any kind of stopping material, either rock dust, ordinary masonry, concrete, or wood. When the pressure is first thrown on the

stopping it will blow out as a mass the inbye side, but if the explosion is not a heavy one, the outbye, or first, set of shelves will remain standing with a certain amount of rock dust on same, to provide material for extinguishing the flame. To protect the rock dust from getting wet, and yet at the same time to permit it to be inspected, separate short cur-

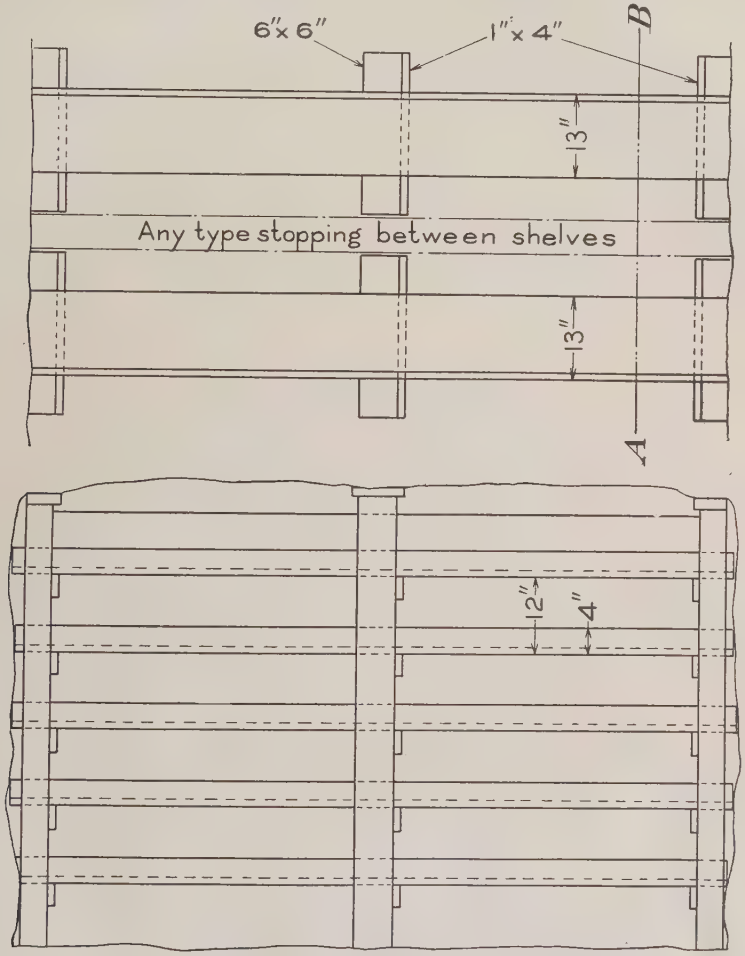


FIG. 13.—ROCK DUST PROTECTED STOPPING FOR MINES.

tains are hung from shelf to shelf. It is not desired to make these in one large curtain, since when the whole stopping is under pressure there is a liability of its going out in a mass, but if the curtains are in narrow strips they will flap up when the pressure wave strikes the stopping barrier.[4]

[4] Patents have been applied for by the author for the foregoing described devices, prospective rights being assigned to the Bureau of Mines, for the benefit of the public.

Future Investigations at the Experimental Mine

Besides the investigation into protective devices, it has become apparent that there is need for testing dusts from coal beds of different districts in order to determine the degree of inflammability or explosibility of the pure coal dust, and then to determine what admixture of roof material, or where artificially ground rock dust is introduced, the amount of the foreign inert material, which will make the coal dust harmless; in other words, to determine the explosive limit of the particular dust (*a*) to prevent ignition, (*b*) to prevent propagation. For example, for Pittsburgh dust it requires 70 per cent. of the roof shale in the mixture to

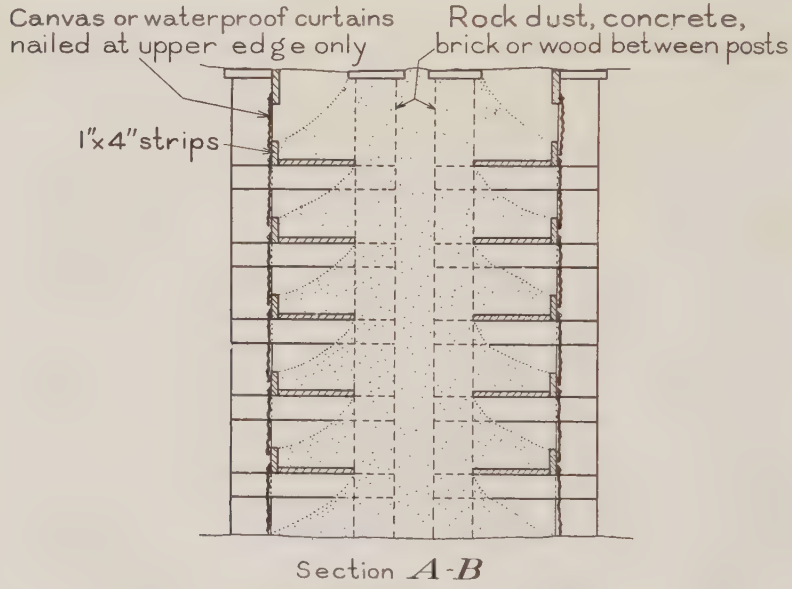

Canvas or waterproof curtains nailed at upper edge only
Rock dust, concrete, brick or wood between posts
1"x4" strips
Section A-B

FIG. 13A.—SECTION ON LINE *A-B* OF FIG. 13.

prevent ignition, but to prevent propagation it requires about 80 per cent. This is when the local shale is used; but when limestone is employed in the mixture, owing to there being no combustible matter in it, 60 per cent. of the mixture will prevent ignition, and 75 per cent., propagation. One low-volatile high-carbon coal which was tested would not propagate an explosion with only 20 per cent. admixture of shale, so that a mine producing coal of this character would apparently be reasonably safe from dust explosions by the introduction of a relatively small quantity of rock dust. On the other hand, this coal in a mixture with 40 per cent. shale propagated an explosion when $1\frac{1}{2}$ per cent. methane was present.

In tests of dusts from other districts at the experimental mine, owing

to the possible admixture of particles of coal abraded from the ribs, which would alter the results, it seemed necessary to line with concrete a portion of the passageways where such tests are conducted, and this is now being arranged at the time of preparing this paper.

One of the most important series that will have to be undertaken is that of determining how much the limit of explosibility of each dust is changed by the introduction of small quantities of firedamp. Some tests have been made, but not enough from which to draw conclusions. Special arrangements are being made for the introduction of gas at the face of one of the entries so that it may be put in behind a diaphragm, then mixed with the air by a fan, in any desired proportions. At the present time it is only feasible to use 2 per cent. or less throughout the whole mine, whereas it is often desirable to make a test with an explosive mixture of gas at the face.

It is hardly necessary to say that the number of kinds of experiments that might be tried is almost infinite, and the work will have to progress slowly, step by step.

Discussion

William Griffith, Scranton, Pa.—I notice in the discussions and literature that emphasis is placed on rock dust, shale dust, and limestone dust, and I infer that these different sorts of rock are ground up to produce this dust. Is that right?

Mr. Rice.—That is correct, but we do not have to grind the limestone dust we are using in the experimental mine, as that is obtained from a quarry, and is used just as it comes; we can get it very cheaply, as it is a by-product of the quarries.

Mr. Griffith.—Have any experiments been made with ordinary dirt out of the fields; could not that be used as ground dust?

Mr. Rice.—If you dry it and powder it enough so it will float in a strong air current it might be all right for the barriers, but if it is damp there will be a tendency for it to mat together; then it would be quite useless for making a dust cloud. There might also be another disadvantage if used through the whole mine, if it had very much free silica in it, which is bad for the lungs. Limestone and selected shale dust would not affect the health of miners.

Mr. Griffith.—In order to do very much good with any of those barriers, they ought to be rather extensively used throughout the mines, and there are few mines that have facilities for grinding up this rock into such a fine dust. If the ordinary earth from a plowed field could be used, even if it were not quite so good, it is more accessible and very much cheaper; therefore, in using it, with the universality of the supply throughout the country, it would tend to stop explosions.

MR. RICE (communication to the Secretary*).—In closing, it is well to again emphasize that I do not advocate barriers being a "cure-all" for explosions. It will be generally agreed that prevention is best, not only by means of improved methods of using explosives, and ventilation to prevent accumulations of firedamp, but also by either thoroughly wetting the dust throughout the mine, or else so treating it with rock dust or other inert dust that an explosion cannot start. From my observations I believe that rock dusting is a practicable method, and has the merit of visibility, and remaining effective for considerable periods, whereas the watering method has to be daily followed up, and if neglected for a short period the mine may become in a dangerous condition. The Germans attribute one of their great disasters to the fact that watering had not been done during the period of a Sunday and a holiday (that is, between Saturday and Tuesday), with consequent drying out of the dust. It is too delicate a matter to refer to specific disasters in this country, but it is thought that many similar instances have occurred in the United States, in mines ordinarily well taken care of, but in which, through temporary neglect of watering, the dust had become dangerous. Where the watering system is used it is undoubtedly a very great help to also sprinkle some rock dust through the entries, which will assist through its capillary attraction in wetting the dust, and will also help in making the dust pack together, so it will not be raised by a concussion.

Referring to Mr. Griffith's remarks, while earth from the fields could doubtless be used if dried, it may be pointed out that to take it from a plowed field would probably destroy ground valuable for agricultural purposes, and it would seem to be cheaper to take material of less value, such as limestone or shale.

Since the original paper was issued one of the largest coal companies is making a test of rock dusting in one of its mines, on stretches of the main haulage entry, and also in one of the butt entries. After first cleaning these roads limestone dust is being used. The conditions are very severe, as the cars are all leaky, and a great deal of coal is spilled off, particularly at the switches. After being down a month, at the time of writing this, according to the samples gathered and the analyses made, it shows most favorably, indicating that the limestone dust is still in sufficient excess to prevent a dust explosion from originating or propagating in these zones. It has the further advantage that the limestone dust is sticking to the ribs and roof so that it has the effect of whitewash, making it easy to illuminate the entry, which in itself is an element of safety. So far the test gives indications that the cost will be less than by the previous method of watering supplemented by calcium chloride; and judging from the rest of the mine it is believed that conditions are much safer where the rock dusting has been done.

*Received Mar. 7, 1915.

Some interesting tests have been conducted at the experimental mine, in which the rock dust is not mixed with the coal dust but the coal dust is strewn on top. It makes a condition of maximum severity for the rock-dusting method. These tests, as yet limited in number, indicate that the rock dust is brought up into suspension and extinguishes the initial explosion nearly as well as if mixed before laying through the entries. The proportion of shale dust to Pittsburgh coal dust must be increased merely from 75 to 77 per cent. in the most extreme case, using wholly fine coal dust 80 per cent. of which passes through a 200-mesh sieve. With coarser coal dust mixed with the finer coal dust, which is the practical condition now under test, it is thought that less than 75 per cent. of shale dust will be needed to neutralize the coal dust spread on top of it.

Coal-Dust Explosion Investigations

BY J. TAFFANEL,* LIEVIN, FRANCE

(Pittsburgh Meeting, October, 1914)

I AM very much impressed by this manifestation of international brotherhood; the mining engineers on both sides of the ocean have similar subjects to deal with, meet with the same difficulties, expose themselves sometimes to similar dangers; they are like companions in arms in the battle fought by mankind in order to subjugate the natural forces and to draw profit therefrom. There is no other field where this fraternity must manifest itself more actively than in the studies pursued with a view to improving the safety of mines and to rendering impossible in the future these mining disasters, which every year claim hundreds of victims. These are very difficult problems, the solution of which requires numerous and laborious experiments accompanied by very arduous scientific investigations; it is extremely desirable that the various mining countries, which have undertaken to solve these difficult problems, should unite their efforts in order to succeed as soon as possible and as well as possible. This idea has been fully grasped by your Bureau of Mines, which has brought me to Pittsburgh; the daily exchange of views with the distinguished chief of the mine investigation division, George S. Rice, and with his excellent collaborators, the comparison of the results obtained at Lievin and Bruceton, the checking of these results by means of new tests, the profound study of the measurements, diagrams, measuring and laboratory apparatus, the co-operative investigation of new apparatus, or the preparation of new investigations, are, I hope, of such a nature as to bring about great progress.

I think that it will be interesting to you to know what has been done in France along this line.

It was the Courrière disaster which led in France, as well as in other countries, to the undertaking of experiments on a large scale with a view to investigating the danger of coal dusts. We have had in France, prior to 1891, numerous explosions in the mines; but beginning from that time, great progress has been made in the fight against firedamp in all the mines where firedamp was found, even if in exceptionally large quantities. To meet this end mine ventilation was considerably improved by modifying, when necessary, the method of working and by substituting, for

* Director of the Government Experiment Station, Lievin, France.

instance, the system of complete filling of the mine excavations for that of partial filling or none at all; use was made of improved safety lamps, particularly of the double-gauze and bonneted Marsaut type; the use of black powder for firing shots in the coal was forbidden; the escape of firedamp was observed with greatest attention by making precise gas analyses every week and sometimes every day in all the splits of the air current. Thanks to these measurements, which have already been made obligatory by the regulation of 1895, not a single large explosion took place between 1891 and 1906, and the proportion of miners killed remained very low, at about one per thousand annually.

But the development of the mines, the deepening of the workings, which became warmer and drier, the increase of the ventilation which dried up the galleries, and many other causes, have increased the dust danger. On March 10, 1906, at the Courrière collieries, which were absolutely free from firedamp, an explosion occurred which devastated the workings of one pit and part of the workings of two neighboring and connected mines and caused the loss of 1,099 lives. This disaster, the greatest that ever occurred in the mines of the world, has demonstrated in an indisputable manner the reality of the coal-dust danger.

Thus was shown in France the imperative necessity of studying this danger and of investigating the means of fighting it. Contrary to what happened in the United States, it was not the government, but the Coal Owners Mining Association of France, that took the initiative in making these investigations and experiments, establishing the station of Lievin, and furnishing all the necessary funds. However, this initiative had the full approval of the government, and led to my appointment as Director, having formerly been a government mining engineer. Further, an administrative committee, the firedamp committee, appointed by the government, keeps in touch with the experiments and ratifies the results.

The first experiments of the experimental station of Lievin were made in 1907 on a restricted scale; but showed that the various kinds of coal dusts had quite different inflammabilities, the most inflammable having the largest proportion of volatile matter, and that by mixing a sufficient amount of stone dust with the coal dust the latter could be made non-inflammable.

From year to year the investigations have been continued on an ever larger scale. At present the testing gallery is 1,000 ft. long and has, at a distance of 750 ft. from the origin, a side branch of 230 ft. The gallery is entirely on the surface; the first 90 ft. had been made of reinforced concrete in order to have a rectangular cross-section; the following portion up to 600 ft. from the origin is circular in cross-section, in order to offer the best form for increasing pressures. It is a steel tube 6 ft. in diameter and $\frac{3}{8}$ in. thick. In one instance an extremely violent dust explosion generated in the last 30 ft. a pressure of about 280 lb. per square

inch and caused the destruction of this part of the gallery. The extensions to 1,000 ft. have been constructed of special steel plates of $\frac{3}{4}$, $\frac{7}{8}$ and 1 in. thickness.

A branch tube connects the gallery with the ventilator, which is used for driving out the smoke after the explosion, and a trap-door protects the ventilator during the majority of the tests; it has been found, in fact, that even if the ventilation should have any slight influence upon the initial phase of the explosion, it does not have any influence on the propagation.

The experimental station also has a smaller steel gallery for the investigation of explosives. The two galleries are so arranged as to receive mixtures of firedamp and air in explosive proportions in an explosive chamber, or with a low content of firedamp in their entire length.

There are several laboratories, one of which is specially devoted to the study of explosives; in addition to various apparatus met with in the majority of laboratories of this kind, there is an extremely rapid cinematograph established last year by the Lievin experimental station, which gives a complete image of the flame of the explosion for every $\frac{1}{1000}$ sec.

The dust explosions produced in the large gallery are investigated with the help of numerous apparatus, chronographs, manometers, gas-sampling bottles, etc., which it would take too much time to describe here; I shall confine myself to explaining a diagram obtained with one of these apparatus. One of the curves, recorded photographically, gives the variation of the pressure as a function of time at the point where the apparatus was located; the first point to be noted is the arrival, at the manometer, of the shock wave produced by the detonation of the explosive; the pressure is maintained by the combustion of the dusts and rises in proporiton as the explosion approaches a maximum. The passage of the flame is photographically recorded on the same film; its passage lasts but a fraction of a second. Another curve gives the velocity of the air; immediately after the passage of the shock wave the air is set in motion in the direction of the explosion, running from the center of combustion; this is the "pioneering" wave which raises the dusts; this first velocity amounts to from 50 to 100 ft. per second under the conditions of this test, but it rises rapidly at the same time as the pressure, and attains a maximum at the moment of passage of the flame; this passage, which takes place a little sooner toward the center of the gallery then on the wall, where the recording is done, has as its effect to reduce considerably the velocity of the air and often during violent explosions to change their direction; the reason thereof is to be found in the increase of volume of the gases in consequence of the combustion; there is, on the one hand, a blast in front of the flame, waves which maintain and reinforce the pioneering wave, and, on the other hand, a recoil of the burned gases

toward the regions where the explosion was less violent and the pressure was lower; this reversion of the direction of the movement of the gases during the passage of the flame is sometimes sufficiently strong to give great dynamic effects in a direction opposite to the direction of the explosion; this explains the contradictions observed sometimes in the investigations in consequence of mine accidents.

An artificial gallery, like the one at Lievin, is easy to clean and to prepare, and allows of performing numerous tests at any season; in certain series of tests we have been able to make one test a day; the total number of tests in the large gallery exceeds at present 1,400. We have, therefore, collected a large number of results. We have investigated the laws of development of the dust explosions, which enabled me to establish in 1910 the theory of explosions based on the laws of combustion and of the dynamics of fluids. Numerous series of tests have been made with the dustless or watered zones, or with the pure-dust zones, or with zones containing variable proportions of stone and coal dust; these zones of from 300 to 600 ft. in length prove to be incapable of stopping a violent explosion. But I succeeded in 1909 and 1910 in stopping explosions, even the most violent, by means of arresting barriers where large masses of water or of stone dust are accumulated at the point where the combustion is to be stopped; these masses are set in action by the pioneering wave. During the last $1\frac{1}{2}$ years, in consequence of an explosion that occurred in the Clarence mine in northern France, which seems to have propagated itself with exceptional slowness, I have been trying to improve the arresting barriers by increasing the quantity of the accumulated extinguishing materials, by securing their setting in action even by weak pioneering waves, and by making their discharge last sufficiently long in the case when the flame arrives several seconds late. I arrived last year at a solution which showed itself to be efficacious by means of a slow-discharging water trough or tank. I have been experimenting for several months on concentrated stone-dust receptacles, which so far have given me good results and will undoubtedly be soon ready to be adopted in practice.

Some hundred tests made between 1908 and 1909 and more than 500 tests made more recently, had as their object to investigate the relative capacity for propagation of various coal dusts mixed with different proportions of stone dust and deposited in variable quantities in the gallery, in the presence of various initial explosions, with or without watering, with or without the presence of various firedamp contents in the atmosphere of the gallery.

At the same time the inflammability of these dust mixtures was measured by means of a laboratory apparatus which was thus calibrated for the gallery tests.

I have thus arrived at an empirical law which, with the help of the "inflammator," a laboratory inflammability-testing apparatus, gives

the relative degree of various dust deposits, according to the value of the multiple factors of the problem.

But there is a factor whose influence it is difficult to determine by means of tests made in a single gallery; it is the influence of the gallery itself, of its sectional area, of the nature of its walls, of the arrangement of the orifices, blind ends, ramifications; the theory indicates that this influence must be considerable. It is therefore essential to make comparative tests in other galleries and to check the results of the artificial gallery by means of tests made in a real mine; only tests made in an experimental mine, like the Bureau's experimental mine at Bruceton, Pa., can be considered as giving results that can be immediately applied to practice. In France we were impressed by this question about two years ago, and I was able to profit last year by the abandonment of a gallery in the Commentry mine, where I made 16 tests which gave results that agree fairly well with the tentative results obtained at Lievin. They showed the efficacy of the arresting barriers and of the new water tanks, and they also showed how bends or turns in the passageways favored the stopping of explosions. The tests were stopped by the destruction of the gallery and by the intrusion of water. In three months I shall have at my disposal a new experimental field in the workings of the Montvicq mine, which will soon be abandoned.

An ideal experimental mine, particularly because it is well equipped with recording apparatus of all kinds and is perfectly adapted to the experimental requirements, is the Bruceton mine, whose tests the Bureau of Mines has kindly permitted me to keep in touch with. I have been extremely interested in the study of the important results obtained so far; I have seen with the most cordial satisfaction the advances made by Mr. Rice. By means of investigations parallel to those pursued at Lievin he has greatly improved the method of stopping explosions, introduced new ideas and made arrangements which seem to be both very practical and very efficacious.

It is not sufficient merely to theorize about the coal-dust danger and the means of overcoming it; it is necessary to introduce in a practical way what is suggested by the experiments.

The majority of the dusty mines of France have made great advances along this line in the course of the last years. The first precautions, which have been taken and which are the most important, are those whose object it is to suppress or prevent the initial causes of inflammation: these are in the first place all the measures which I have indicated in the first part of my remarks relative to the firedamp danger; also, the extension of the safety explosives to the dusty mines, the suppression in these mines of lamps with naked flames, the assignment of special employees to the shot firing, the recommendation of firing the shots preferably after or between shifts. The most important point after that consists in

rendering the dust deposits as little inflammable as possible over the entire length of the haulage roadways and of the principal arteries of the mine; this neutralization of the dust is rarely done through watering, almost always through removal of the dust followed by stone or rock dusting. Practice has shown that, on condition that tight cars are used, the result of such an operation is to maintain the proportion of ash above 60 or even above 70 per cent. for two, three months, or sometimes longer; if the inflammability tests which are made regularly in the majority of the dusty mines show that the ash content must be maintained above 65 per cent., the operation must be renewed almost every three months; with less inflammable coals and a less elevated limit of the ash content, the operations of neutralization could be made at longer intervals.

Finally, as a third precaution, the mine sections are separated by arresting barriers; about 2,000 of these devices are actually installed in the French mines.

Such is the organization against the dust danger, which is on the way of being put into practice in all the dusty mines of France, and which has already been put into practice where the nature of the dusts or the presence of firedamp rendered these measures more necessary.

Coal-Mine Explosions Caused by Gas or Dust

BY HOWARD N. EAVENSON, GARY, W. VA.

(Pittsburgh Meeting, October, 1914)

In a discussion in the *Transactions* of the Institute (vol. xl, page 835 *et seq.*) the writer gave some data about explosions of gas and dust in the coal mines of the United States, Canada, and Mexico, among which was a table showing all explosions in the countries mentioned, in which five or more fatalities occurred, which had been reported from Oct. 2, 1871, to Mar. 28, 1908. Since the date of publication of this discussion, as opportunity has presented, this table has been extended by the addition of explosions occurring since that time, by including several explosions which occurred prior to 1871, and by adding to the list all explosions of gas or dust in which less than five fatalities occurred, this having been done by a search through the various mine reports, partly by the author and partly by the Librarian of the United Engineering Societies, and by correspondence with mine inspection officials of the various States, to all of whom acknowledgment of assistance is made. The accompanying list, Table I, is believed to be as nearly correct as such a statement can be made, and, while only extending to Dec. 20, 1912, contains all of the data now available in reports.

An analysis of Table I, showing the occurrence by months of all explosions of gas or dust, including those in which less as well as those in which more than five fatalities occurred, is given in Table II.

The statements are frequently made, in technical papers, that the number of explosions, particularly the larger ones, is increasing more rapidly than the number of mines, and that the number of fatalities caused by mine explosions, particularly the more disastrous ones, is increasing faster, proportionally, than the production of coal. To investigate the correctness of these statements Table III was prepared, in which the number of accidents and fatalities is taken from Table I, using those in the United States only, on account of lack of other data for Canada and Mexico; the production is from the records of the U.S. Geological Survey, and the number of mines, from 1909, from the same source; previous to 1909, the number of mines, for the decennial periods, was furnished by the U. S. Census records.

It had been the writer's intention to prepare similar data for European mines, but after considerable correspondence and labor this has been found

impossible without convenient access to the records, which he has been unable to have. From the files of the *Colliery Guardian*, and from the editor of that paper, a great deal of data about British explosions was obtained; for the years 1889 to 1904, inclusive, the tables showing maximum and minimum temperatures, barometer readings, etc., published in the *Proceedings of North of England Institute of Mining and Mechanical Engineers*, vol. xxix, *et seq.*, gave complete data for all explosions; those between 1658 and 1841, inclusive, were obtained from Appendix E, Report of South Shields Committee to Investigate the Causes of Accidents in Coal Mines, by James Mather, Secretary.

The complete list is contained in Table IV. It is believed to be correct for those explosions in which more than five fatalities occurred, but is incomplete for those in which less than this number were killed.

An analysis of Table IV, showing the occurrence of the various kinds of explosions by months, is shown in Table V.

Through the courtesy of J. Taffanel, of Lievin, France, Mr. Lemaire prepared for the writer a list of the recorded explosions in French and Belgian mines, that for the French mines covering the period from 1814 to 1903, inclusive, no later data being available, and for the Belgian mines from 1887 to 1909, inclusive.

The list of French explosions is given in Table VI, and the analysis showing the occurrence by months of the various kinds of explosions in Table VII.

The list of explosions in Belgian mines is given in Table VIII, and the analysis showing occurrence by months in Table IX.

For more convenient study, the data shown in Table III, for the years since 1870, when our modern records may be said to begin, have been grouped into five-year periods, the results being shown in Table X. Too much weight must not be attached to the "Accidents per mine" columns, as prior to 1909 the number of mines is obtained from census reports and it is evident that a "mine" did not have the same meaning in each case, as the difference in number of mines reported in 1880 and in 1889 will show. The number of mines for each five-year period, of course, is interpolated from the data in Table III.

Conclusions

From the data given, the following conclusions are evident:

1. In North America, minor explosions, or those in which less than five fatalities occurred, happen most frequently in October, November, December, January, and March, although nearly as many have happened in June as in March; those in December, January, and February are above the average in fatalities, as are also those in May and July, those occurring in May having a slightly greater average fatality than even those of December.

Serious explosions, or those in which more than five lives have been lost, have happened most frequently in January, February, March, April, and November. Contrary to the usual belief, the number of explosions in December has been slightly below the average, although their intensity, and the number of lives lost, have been considerably greater than those of any other month. January, February, and May are also above the average in the number of fatalities per explosion, May being next to December in this respect.

For all explosions of gas or dust, January, March, November, October, and December, in the order named, are above the average in *number* of explosions; in number of fatalities per explosion, May, December, February, January, and March are above the average. For all explosions, therefore, May exceeds any of the winter months in number of fatalities per explosion.

2. In the coal mines of the United States, the total number of accidents and of fatalities due to explosions of gas or dust has been steadily increasing; there has been a slight increase in the number of accidents and a more decided increase in the number of fatalities per million tons produced; the serious explosions, causing five or more fatalities each, have been steadily increasing in actual number and number of fatalities, as well as in number of accidents and of fatalities per million tons produced; the number of accidents per mine, both serious and total, also shows a steady increase. It is therefore true that we have more explosions, and more serious ones, both actually and in relation to our number of mines and production, than we had years ago.

3. In the coal mines of Great Britain, minor explosions, so far as our records show, have occurred most frequently in the months of August, October, May, February, September, and December, while those in April, March, May, August, and November are above the average in number of fatalities per explosion. Serious explosions have occurred most frequently in December, October, November, and March, while those in June, December, July, February, and May have been of more than average intensity. For all explosions, August, October, December, February, and May are above the average in number, and December, June, July, February are above the average in intensity. By far the largest number of fatalities has occurred in December, June being second in this respect.

4. In the coal mines of France, from 1814 to 1904, minor explosions have occurred most frequently in July, August, February, April, May, and January and have been of more than average intensity in April, December, July, February, August, and October. Serious explosions have been above the average in frequency in August, April, July, and March, and in intensity in January, September, December, July, November, and October. For all explosions, July, August, April, and May are above the

average in number, and December, January, November, July, March, October, and September in intensity. By far the largest number of both accidents and fatalities have occurred in July.

5. In the coal mines in Belgium, 1891 to 1909, minor explosions occurred more frequently than the average in June, May, and July, and in March, January, February, April, and June were of more than average intensity. Serious explosions occurred most frequently in May, March, and July and were of more than average intensity in March. For all explosions, May, July, and June are above the average in number and March in intensity. By far the largest number of fatalities occurred in March.

TABLE I.—*Mine Explosions Supposed to be Caused by Gas or Dust, Officially Reported in North America*

Date	Place	Number of Fatalities
1839–March 18,	Black Heath, Va	40
1854–May 15,	Chesterfield, Va	19
1855–May 19,	Midlothian, Va	42
1869–Aug. 6,	Wadlingers, Pa	1
Dec. 13,	Mine Hill Gap, Pa	1
1870–Feb. 11,	Norwegian Mine, Pa	1
May 28,	Hein & Glasmire	1
Oct. 18,	Phoenix No. 3, Pa	1
Oct. 29,	St. Clair Mine, Pa	1
Dec. 13,	Beechwood, Pa	1
Dec. 18,	Beechwood, Pa	1
1871–Jan. 4,	Forestville, Pa	2
Jan. 4,	Ravensdale, Pa	1
Jan. 22,	Revenue, Pa	1
April 14,	Big Lick, Pa	1
April 15,	N. Philadelphia Mine, Pa	1
June 2,	Live Oak Mine, Pa	2
June 18,	Feeder Dam Mine, Pa	1
July 22,	Swift Creek Mine, Pa	1
Sept. 12,	Pine Forest, Pa	1
Sept. 26,	Buckville, Pa	2
Oct. 2,	Otto Red Ash, Pa	5
Oct. 3.	Otto Red Ash, Pa	1
Oct. 14,	Beechwood, Pa	2
1872–Jan. 1,	Montilius, Pa	1
April 10,	Thomaston, Pa	1
April 17,	Locust Run, Pa	2
April 25,	Otto Red Ash, Pa	2
Aug. 14,	Lower Ranch Creek, Pa	2
Sept. 1,	East Pine Knot Mine, Pa	1
Sept. 1,	St. Clair Shaft, Pa	1
Sept. 27,	Diamond, Pa	7
Oct. 14,	Otto White Ash, Pa	1
Nov. 8,	Otto White Ash, Pa	2
Dec. 12,	Daniel Webster Mine, Pa	1
1873–Jan. 28,	St. Clair Shaft, Pa	2
Feb. 27,	Daniel Webster Mine, Pa	1
April 29,	New Kirk, Pa	1
May 13,	Intercolonial Coal Co., N. S.	55
June 8,	Glen Dower, Pa	1
June 10,	Henry Clay, Shamokin, Pa.	10
Sept. 6,	Eagle Hill Mine, Pa	1
1873–Oct. 10,	Gilberton Mine, Pa	1
Oct. 11,	Gilberton Mine, Pa	1
Nov. 10,	Plank Ridge Mine, Pa	1
Nov. 26,	Honey Brook Mine, Pa	1
1874–April 12,	Big Mine Run, Pa	1
April 14,	Anchor Mine, Pa	1
June 2,	Tunnel No. 8 Mine, Pa	1
June 8,	Ravensdale Mine, Pa	1
June 23,	Beechwood Mine, Pa	3
July 27,	Anchor Mine, Pa	2
Sept. 2,	Shenandoah City, Pa	1
Sept. 26,	Treverton Mine, Pa	1
Sept. 28,	Anchor Mine, Pa	1
Oct. 23,	Big Mountain Mine, Pa	1
Oct. 26,	Mariam Mine, Pa	1
Nov. 2,	Buckville, Pa	1
Nov. 10,	Mine Hill Gap Mine, Pa	2
Dec. 1,	Girard, Pa	1
1875–Sept. 30,	Anchor Mine, Pa	3
Sept. 30,	Hickory Shaft Mine, Pa	1
Oct. 12,	Keystone Mine, Pa	1
Dec. 5,	Ellangoan Mine, Pa	1
Dec. 12,	Hickory Shaft Mine, Pa	1
Dec. 13,	Hickory Shaft Mine, Pa	1
1876–Jan. 20,	Forestville, Pa	1
Jan. 28,	Parsons Spring, Pa	1
Feb. 12,	Exeter, West Pittston, Pa	4
Feb. 23,	Black Mine, Pa	1
March,	Exeter, West Pittston, Pa	1
May 20,	Midlothian, Va	8
June 2,	Phoenix No. 2, Pa	1
June 6,	Phoenix No. 2, Pa	1
June 8,	Phoenix No. 2, Pa	1
June 20,	Phoenix No. 2, Pa	1
July 24,	Black Diamond, Cal	6
Oct. 24,	Anchor, Pa	1
Oct. 31,	Hickory Shaft, Pa	1
Nov. 6,	Hickory Shaft, Pa	1
Nov. 20,	Plymouth, Pa	1
Dec. 2,	Henry Colliery, Pa	1
1877–March 16,	Mine Hill Gap, Pa	2
June 9,	Wadesville, Pa	7
1878–Jan. 15,	Potts Colliery, Pa	5

Mine Explosions—Continued

Date	Place	Number of Fatalities
1878–Feb. 28,	Preston Colliery, Pa	1
April 8,	Keystone, Pa	2
May 21,	Sydney, N. S	5
June 15,	Short Mt. Slope, Pa	1
Nov. 15,	Merriam Colliery, Pa	1
Nov. 21,	Sullivan, Ind	8
1879–Jan. 1,	Beechwood, Pa	1
March 4,	Hausch Creek, Pa	2
May 28,	Wyoming, Pa	1
June 20,	Mill Creek, Pa	5
July 21,	Beechwood, Pa	1
Oct. 2,	Thomaston, Pa	1
Nov. 6,	Audenreid, Pa	6
1880–April 8,	Preston No. 2, Pa	1
May 3,	Henry Clay Shaft, Pa.	1
May 3,	Lykens Valley Slope, Pa	5
Nov. 27,	Cameron, Pa	1
1881–Jan. 8,	Diamond Colliery, Pa	1
Jan. 14,	Twin Shaft, Pittston, Pa	1
Feb. 10,	Robbins, Ohio	6
March 4,	Almy, Wyoming	38
March 5,	Shaft No. 2, Nanticoke, Pa.	6
April 11,	Van Storch Slope, Providence, Pa	1
May 7,	Central Shaft, Hyde Park, Pa	1
Sept. 5,	No. 4 Shaft, Pa	2
1882–Feb. 3,	Midlothian, Va	32
March 25,	Laurel Run Slope, Pa	1
May 24,	Kohinoor Colliery, Pa	5
June 15,	Stanton Air Shaft, Pa	2
July 31,	West Penna., Pa	1
Sept. 5,	Wyoming Shaft, Pa	1
Sept. 6,	Conyngham, Pa	2
Sept. 25,	Dodson Shaft, Pa	2
Oct. 27,	Pine Ridge Shaft, Pa	2
Nov. 20,	Mahanoy City, Pa	1
1883–Jan. 9,	Coulterville, Ill	10
Jan. 15,	Packer No. 4, Pa	3
Feb. 26,	Mt. Pleasant Slope, Scranton, Pa	1
March 12,	W. Penn, Pa	1
March 22,	Packer No. 2, Pa	2
April 19,	Keystone Colliery, Pa	4
May 4,	Dorrance, Pa	2
May 9,	No. 2 Shaft, Nanticoke, Pa.	1
May 21,	Mineral Spring, Pa	2
May 23,	Cameron Colliery, Pa	1
May 24,	Fair Lawn Slope, Scranton, Pa	3
June 5,	Luke Fidler Colliery, Pa	1
July 23,	No. 1 Shaft, Nanticoke, Pa	1
Oct. 29,	No. 7 Shaft, Pa	2
Nov. 5,	Enterprise, Pa	1
Dec. 13,	Eagle Shaft, Pittston, Pa	1
1884–Jan. 24,	Crested Butte, Colo	59
Feb. 21,	Leisenring, No. 2, Pa	19
March 13,	Pocahontas, Va	114
April 14,	Lovedale, Pa	2
May 30,	Tunnel, Pa	1
Aug. 20,	Greenback, Pa	7
Oct. 24,	Youngstown, Pa	14
1884–Dec. 8,	Henry Clay No. 1, Pa	4
1885–Feb. 2,	Savanna, I. T	6
Feb. 17,	Hillman Vein, Pa	2
Feb. 18,	Vale Colliery, N. S	13
June 4,	No. 4 Slope, Nanticoke, Pa.	3
Oct. 21,	Plymouth, Pa	6
1886–Jan. 13,	Almy, Wyoming	13
Jan. 21,	Newburg, W. Va	39
March 8,	Uniondale, Pa	7
Aug. 30,	Fair Lawn Colliery, Pa	5
Oct. 28,	Greenridge, Pa	1
Nov. 26,	Conyngham, Wilkes-Barre, Pa	12
1887–April 2,	Slope No. 3, Luzerne Co., Pa	1
April 4,	Savanna, I. T	18
June 23,	Slope No. 4, Luzerne Co., Pa	3
Oct. 26,	Nottingham, Pa	1
1888–March 29,	Rich Hill, Mo	26
Nov. 5,	Starkville No. 2, Colo	2
Nov. 7,	Kettle Creek, Pa	17
Nov. 9,	Frontenac, Kan	40
Dec. 3,	Newcastle, Colo	3
Dec. 10,	Coal Creek No. 2, Colo	2
1889–Jan. 23,	Slope No. 4, Nanticoke, Pa.	2
Feb. 5,	Shaft No. 1, Edwardsdale, Pa.	1
April 14,	No. 3 Colliery, W. Nanticoke, Pa	2
April 23,	Shaft No. 2, Nanticoke, Pa.	1
May 10,	Powers Mine, Pa	4
June 10,	Greenridge Slope, Pa	1
July 24,	Central Shaft, Scranton, Pa.	2
Aug. 22,	Olyphant No. 2, Pa	4
Sept. 2,	Jersey No. 8, Ashley, Pa	1
1890–Jan. 29,	Como No. 5, Colo	1
Feb. 1,	Nottingham Colliery, Pa	8
Mar. 3,	Shaft No. 3, S. Wilkes-Barre, Pa	8
April 2,	Slope No. 4, Nanticoke, Pa.	5
May 15,	Jersey No. 8, Ashley, Pa	28
May 17,	Empire, Pa	2
July 25,	Lincoln, Ill	1
Sept. 14,	Shaft No. 4, Edwardsdale, Pa	1
1891–Jan. 2,	Staunton, Ill	1
Jan. 3,	Centralia, Pa	1
Jan. 14,	Centralia No. 1, Nanticoke, Pa	2
Jan. 14,	Luke Fidler, Pa	2
Jan. 15,	Bast, Pa	1
Jan. 15,	Lincoln, Ill	2
Jan. 21,	Marissa, Ill	3
Jan. 27,	Mammoth, Pa	109
Feb. 21,	Springhill Mines, N. S	125
May 7,	Ocean, W. Va	4
May 8,	No. 10 Colliery Lehigh C. & N. Co., Pa	4
Aug. 3,	West Fairmont, W. Va	1
Aug. 20,	Nottingham, Pa	1
Oct. 2,	York Farm, Pa	1
Oct. 8,	Richardson, Pa	7

Mine Explosions—Continued

Date	Place	Number of Fatalities
1891–Oct. 16,	Merriam, Pa	1
Oct. 24,	Coal Ridge Mine, Colo	2
Nov. 5,	New Boston, Pa	1
Nov. 8,	Shaft No. 1, Nanticoke, Pa.	12
Dec. 26,	Cameron, Pa	1
1892–Jan. 4,	Primrose, Pa	1
Jan. 7,	Krebs, Ind. Ter	67
Jan. 18,	Glendon, Pa	3
Jan. 22,	Packer No. 2, Pa	1
Jan. 30,	Shaft No. 1, Lee Vein, Pa	3
Feb. 20,	Packer No. 3, Pa	2
March 9,	Alden, Pa	3
May 10,	Roslyn, Wash	45
May 31,	West End, Pa	4
July 13,	Hillman Vein, Pa	1
July 13,	Jersey No. 8, Pa	4
July 14,	Shaft No. 1, Edwardsdale, Pa	2
July 23,	York Farm, Pa	15
Sept. 28,	Wolsen Mine, Colo	2
Oct. 4,	Plymouth, Pa	1
Oct. 8,	Shaft No. 2, Nanticoke, Pa.	3
Nov. 8,	Pekay Mine, Iowa	3
Dec. 21,	Parrish, Pa	1
Dec. 27,	Avondale, Pa	3
1893–Jan. 10,	Como, Colo	24
Feb. 14,	Cedar Mine, Iowa	8
March 2,	Glen Lyon, Pa	1
March 27,	Shaft No. 1, Nanticoke, Pa.	6
April 5,	Shaft No. 4, Edwardsville, Pa	6
April 7,	Big Mt. Mine, Tenn	5
June 22,	Susquehanna No. 1, Nanticoke, Pa	5
Sept. 19,	Dorrance, Pa	1
Sept. 21,	Plymouth, Pa	6
Oct. 20,	Shaft No. 3, Edwardsdale, Pa	1
Dec. 22,	Krebs No. 11, Ind. Ter	3
Dec. 26,	Nottingham, Pa	1
1894–Feb. 20,	Indian Ridge, Pa	1
March 7,	Hillman Vein, Pa	1
March 15,	Lawrence, Pa	1
March 20,	Nottingham, Pa	2
May 1,	Packer No. 5, Pa	2
May 9,	Wimstown, Pa	1
May 10,	Merriam, Pa	1
May 14,	Bear Ridge, Pa	1
May 14,	W. Penn, Pa	3
May 28,	Nottingham, Pa	1
June 23,	Girard, Pa	2
July 16,	Wimstown, Pa	2
July 16,	Henry Clay, Pa	1
July 24,	Stanton, Pa	1
July 25,	Phœnix No. 1, Ohio	1
Aug. 16,	North Nanticoke, Pa	1
Aug. 18,	Maple Hill, Pa	2
Aug. 24,	Gilberton, Pa	1
Aug. 24,	Shenandoah, Pa	1
Oct. 9,	Coal Creek Mine, Wash	4
Nov. 20,	Blanche, W. Va	8
Nov. 27,	Jack Oak Mine, Colo	1
1894–Dec. 3,	Dorrance, Pa	1
1895–Jan. 22,	Sturgis, Ky	5
Feb. 18,	Bear Ridge, Pa	5
Feb. 21,	West Penn, Pa	1
Feb. 27,	White Ash Mine, N. M	24
March 4,	Packer No. 4, Pa	1
March 20,	Red Canyon, Wyo	60
April 8,	Blue Canyon, Wash	23
April 17,	South No. 1 Shaft, Nanticoke, Pa	1
May 10,	Sopris Mine, Colo	3
July 6,	Glen Lyon, Pa	1
July 9,	T. C. I. & R. R. No. 1, Ala.	1
Aug. 16,	Hammond, Pa	1
Oct. 7,	Dorrance, Pa	7
Oct. 19,	Knickerbocker, Pa	1
Oct. 31,	Hillman Vein, Pa	1
Dec. 12,	Indian Ridge, Pa	1
Dec. 19,	Cumnock, N. C	39
Dec. 20,	Dayton, Tenn	25
1896–Feb. 18,	New Castle, Colo	49
March 23,	Berwinsdale, Pa	15
March 25,	Shaft No. 1, Edwardsdale, Pa	1
April 1,	Shaft No. 9, Sugar Notch, Pa	2
April 27,	Alderson, Ind. Ter	1
May 10,	Glenwood, Ohio	1
May 26,	Alderson, Ind. Ter	1
June 8,	Alderson, Ind. Ter	2
July —,	Woodward, Pa	1
Aug. 11,	Dodson, Pa	1
Aug. 15,	Franklin, Pa	1
Aug. 19,	Baltimore No. 2, Wilkes-Barre, Pa	1
Oct. 29,	Shaft No. 3, S. Wilkes-Barre, Pa	6
Dec. 26,	Princeton, Ind	7
1897–Jan. 4,	Alderson, Ind. Ter	5
Feb. 12,	Stanton, Pa	1
March 3,	Middle Creek, Pa	2
March 3,	William A., Pa	2
March 9,	Dickson, Pa	1
March 22,	Lytle, Pa	1
March 23,	W. Penn, Pa	2
March 28,	Jermyn No. 1, Pa	5
April 16,	Monarch Coal Co., Ky	2
May 5,	Buttonwood, Pa	1
May 7,	Auchincloss, Pa	2
June 12,	Phœnix Shaft, Duryea, Pa	1
June 24,	Marion, Pa	2
July 20,	Gowen, Ind. Ter	1
July 26,	St. Nicholas, Pa	1
Aug. 25,	Shaft No. 1, Edwardsdale, Pa	1
Sept. 3,	Sunshine, Colo	12
Sept. 23,	Packer No. 4, Pa	3
Sept. 23,	Shenandoah, Pa	1
Sept. 27,	Alderson, Ind. Ter	1
Oct. 2,	Pettibone Shaft, Pa	2
Oct. 5,	Parrish, Pa	3
Oct. 18,	Harry E. Shaft, Pa	1

Mine Explosions—Continued

Date	Place	Number of Fatalities
1897–Oct. 14,	Kaska William, Pa	4
Nov. 16,	Parrish, Pa	1
Dec. 6,	Clear Spring Shaft, Pa	1
Dec. 27,	Bellevue Shaft, Pa	1
1898–Jan. 5,	Avondale, Pa	3
Jan. 17,	Shaft No. 4, Plymouth, Pa.	2
Jan. 28,	Alden, Pa	1
Feb. 9,	Dodson, Pa	2
Feb. 12,	Gowen, I. T	1
Feb. 14,	Woodward No. 1, Pa	1
Feb. 28,	Alderson, I. T	2
Feb. 28,	Rockwood Mine, Tenn	1
March 19,	T. C. I. & R. R. No. 2, Ala	6
March 22,	Manown, Pa	1
May 26,	Hargrove, Ala	2
June 15,	Henry E. Shaft, Pa	1
June 16,	Parrish, Pa	1
July 5,	Glen Lyon, Pa	1
July 5,	Seneca Slope, Pa	1
Aug. 3,	Williamston, Pa	2
Aug. 15,	Newcastle, Ala	2
Aug. 19,	Hartshorne, I. T	1
Aug. 19,	Maxwell, Pa	1
Aug. 26,	Shaft No. 1, Nanticoke, Pa.	1
Sept. 15,	Shaft No. 2, Nanticoke, Pa.	3
Sept. 21,	Cameron, Pa	1
Sept. 23,	Wilburton, I. T	1
Sept. 23,	Brownsville, Pa	8
Oct. 17,	Forty Fort, Pa	1
Nov. 8,	Schovley Shaft, Pa	1
Nov. 26,	Preston No. 3, Pa	1
Dec. 6,	No. 4 Shaft, Pa	1
Dec. 7,	Luke Fidler, Pa	3
Dec. 15,	Rush Run, W. Va	1
Dec. 19,	Hollenback, Pa	1
Dec. 23,	Exeter, Pa	1
Dec. 24,	Luke Fidler, Pa	2
1899–Jan. 17,	Shaft No. 5, Plymouth, Pa.	1
Jan. 25,	Gilberton, Pa	2
Jan. 28,	Shaft No. 1, Edwardsdale, Pa	2
Feb. 21,	Blocton No. 2, Ala	5
March 1,	Shaft No. 5, Plymouth, Pa.	1
March 9,	Mahanoy City, Pa	8
March 13,	Turkey Run, Pa	1
April 21,	Madrid, New Mexico	5
April 22,	Ola, I. T	2
April 27,	Woodward No. 2, Pa	1
May 3,	Maple Hill, Pa	1
May 7,	Morgan Slope, Wash	1
May 23,	Cumnock, N. C	23
June 16,	Dominion Colliery No. 4, N. S	11
June 22,	Maxwell, Pa	1
July 24,	Shaft No. 4, Edwardsdale, Pa	1
July 24,	Grindstone, Pa	5
Aug. 17,	Shenandoah City, Pa	1
Aug. 21,	Red Star, W. Va	1
Sept. 1,	Gaston, W. Va	1
Oct. 9,	Elm Grove, W. Va	1
Oct. 23,	Nottingham, Pa	1
1899–Oct. 23,	Stanton, Pa	1
Oct. 26,	Alderson, I. T	1
Dec. 9,	Carbonado, Wash	31
Dec. 15,	Ola, I. T	1
Dec. 23,	Sumner, Pa. (Braznell)	19
1900–Jan. 15,	No. 3, So. Wilkes-Barre, Pa.	2
Jan. 22,	Marvine, Pa	1
Feb. 19,	Bottom Creek, W. Va	1
Feb. 21,	Black Diamond, No. 14, Wash	1
March 6,	Red Ash, W. Va	46
March 9,	Pancoast, Pa	1
March 21,	Richmond No. 3, Pa	1
March 26,	Packer No. 3, Pa	2
March 26,	Hartshorne, I. T	1
May 1,	Scofield, Utah	200
June 4,	Warrior Penn, Pa	1
June 7,	Royal Oak, Pa	1
June 8,	Phœnix No. 2, Ohio	3
June 12,	Williamston, Pa	2
July 9,	Maxwell No. 20, Pa	1
July 31,	No. 5, So. Wilkes-Barre, Pa.	2
Aug. 11,	No. 1 Shaft, Lackawanna, Pa	1
Aug. 27,	McAlester, I. T	2
Sept. 6,	Stanton, Pa	1
Oct. 3,	Shaft No. 1, Kingston, Pa	4
Oct. 10,	Panama, I. T	2
Oct. 22,	Witterville, I. T	1
Nov. 1,	Marvine, Pa	2
Nov. 2,	Berryburg, W. Va	15
Nov. 5,	Hartshorne, I. T	1
Nov. 7,	Alderson, I. T	1
Nov. 9,	Buck Mountain, Pa	7
Nov. 30,	Maxwell, Pa	1
Dec. 5,	Luke Fidler, Pa	2
Dec. 6,	Krebs, I. T	1
Dec. 14,	McAlester, I. T	1
Dec. 18,	Henry Clay, Pa	1
1901–Jan. 3,	Laurel Run Slope, Pa	3
Jan. 3,	Delaware Shaft, Pa	1
Jan. 3,	No. 2 Colliery, Plymouth, Pa	1
Jan. 5,	Cleveland No. 4 Mine, Iowa	2
Jan. 27,	Coal City, Ill	1
Feb. 15,	Union, B. C	64
Feb. 28,	Krebs, I. T	3
March 5,	Stanton, Pa	2
March 25,	Gates, Pa	4
March 26,	Wanamie No. 18, Pa	1
April 9,	Wanamie No. 18, Pa	1
April 29,	Alderson, I. T	6
May 15,	Chatham, W. Va	10
May 20,	Richland, Tenn	20
May 27,	No. 4 Shaft, Kingston, Pa	1
June 10,	Port Royal, Pa	19
July 2,	Lookout Shaft, Pa	2
July 29,	Wilburton, I. T	1
Aug. 16,	Lance No. 11, Pa	1
Aug. 29,	Dow, I. T	1
Aug. 30,	Sugar Notch No. 9, Pa	1
Sept. 11,	Stevens Slope, Pa	1

Mine Explosions—Continued

Date	Place	Number of Fatalities
1901–Sept. 16,	Spring Gulch, Colo	6
Oct. 2,	Harry E. Shaft, Pa	1
Oct. 7,	Hartshorne, I. T	2
Oct. 25,	Buttonwood, Pa	6
Oct. 30,	Twin No. 1 Shaft, Luzerne Co., Pa	1
Oct. 31,	Parrish, Pa	2
Nov. 11,	Labelle, Ohio	3
Dec. 3,	Grindstone, Pa	3
Dec. 20,	Ola, I. T	4
1902–Jan. 4,	Cameron, Pa	2
Jan. 23,	Twin Shaft, No. 2, Luzerne, Pa	3
Jan. 24,	Lost Creek, Iowa	20
Jan. 25,	No. 4 Shaft, Luzerne, Pa	1
Jan. 31,	Henry Clay, Pa	1
Feb. 18,	Hocking Mine, Iowa	2
Feb. 26,	Haileyville, I. T	2
March 6,	Catsburg, Pa	5
March 31,	Dayton, Tenn	16
April 8,	Wyoming Shaft, Pa	1
May 19,	Coal Creek (Fraterville), Tenn	184
May 20,	Colgate, I. T	2
May 22,	Fernie, B. C	125
July 10,	Rolling Mill Mine, Johnstown, Pa	112
July 27,	McCurtain, I. T	2
Aug. 7,	Bowen, Colo	13
Sept. 2,	Mt. Lookout, Pa	1
Sept. 16,	Algoma, W. Va	17
Sept. 22,	Stafford, W. Va	6
Oct. 1,	Lawson, Wash	11
Nov. 2,	Locust Run, Ohio	3
Nov. 29,	Luke Fidler, Pa	7
Dec. 8,	Wilburton, I. T	1
Dec. 27,	Little Redstone, Pa	4
1903–Jan. 6,	Alderson, I. T	1
Jan. 10,	Hartshorne, I. T	1
Jan. 13,	Bliss, Pa	1
Jan. 14,	Packer No. 4, Pa	2
Jan. 14,	Taylor Mine, Pa	1
Feb. 5,	Old Forge No. 1, Pa	1
Feb. 25,	Mahanoy City, Pa	1
March 12,	Wilburton, I. T	2
March 13,	Livingston, Ill	3
March 14,	Warrior Run, Pa	1
March 15,	Cardiff, Ill	5
April 13,	Carbon, I. T	6
April 16,	Carbon, I. T	3
May 12,	Ellangowan, Pa	1
May 18,	Wilburton, Pa	1
May 21,	Pooz Mine, W. Va	1
May 21,	No. 14 Shaft, Luzerne Co., Pa	1
May 29,	Bliss, Pa	1
June 17,	Warrior Run, Pa	2
June 19,	Blossburg, N. Mex	5
June 22,	Warrior Run, Pa	1
June 22,	Lehigh C. & N. Co. No. 8, Pa	1
June 29,	Clear Spring, Pa	1
1903–June 30,	Hanna, Wyo	169
June 30,	Colgate, I. T	1
July 2,	Storrs No. 1, Pa	3
July 15,	Union, B. C	16
Sept. 30,	Avondale, Pa	1
Oct. 6,	So. Wilkes-Barre, Pa	1
Oct. 9,	Haileyville, I. T	1
Nov. 18,	Lehigh C. & N. Co. No. 10, Pa	1
Nov. 21,	Ferguson, Pa	17
Nov. 24,	Century, W. Va	3
Nov. 24,	Henry, W. Va	1
Nov. 28,	Stanton, Pa	1
Dec. 3,	No. 14 Shaft, Luzerne Co., Pa	4
Dec. 12,	Wilburton, I. T	1
Dec. 18,	Big Mountain, Pa	1
Dec. —,	Flat Top, Ala	5
Dec. 21,	Nottingham, Pa	1
Dec. 23,	Exeter, Pa	1
Dec. 30,	Sutter, Pa	2
1904–Jan. 5,	McCurtain, I. T	2
Jan. 5,	Auchincloss, Pa	1
Jan. 8,	Crows Nest, B. C	7
Jan. 15,	Corbin, Pa	3
Jan. 25,	Harwick, Pa	178
Jan. 25,	Foster Mine, Iowa	2
Feb. 6,	Storrs, Pa	2
Feb. 8,	Buck, I. T	2
Feb. 12,	McAlester, I. T	1
Feb. 24,	Hillman Slope, Pa	1
March 9,	Tunnel Ridge, Pa	2
March 14,	Tunnel Ridge, Pa	1
March 15,	Henry No. 33, W. Va	2
April 6,	No. 14 Shaft, Luzerne Co., Pa	1
April 17,	Nixon, Pa	1
April 20,	Stearns, Ky	5
June 20,	Maxwell, Pa	1
June 25,	Kangley, Ill	1
Aug. 5,	Pittsburg & Eastern No. 2, Pa	1
Aug. 19,	Short Mt., Pa	1
Aug. 20,	Burnside, Pa	1
Aug. 23,	Buck Ridge, Pa	1
Sept. 1,	Henry Shaft, Pa	1
Sept. 17,	Primrose, Pa	3
Sept. 22,	No. 14 Shaft, Luzerne Co., Pa	1
Sept. 28,	Mt. Jessop, Pa	1
Sept. 30,	Sutter, I. T	1
Oct. 26,	Oxford, Lackawanna Co., Pa	4
Oct. 28,	Tercio, Colo	19
Nov. 10,	Krebs, I. T	2
Nov. 15,	Adkins, I. T	1
Nov. 18,	Crows Nest, B. C	14
Nov. 25,	Wilburton, I. T	2
Dec. 7,	Burnett, Wash	17
Dec. 8,	Penna. Mine, Northumberland Co., Pa	1
Dec. 16,	McAlester, I. T	1

Mine Explosions—Continued

Date	Place	Number of Fatalities
1904–Dec. 19,	Oneida No. 1, Pa	2
Dec. 29,	Alderson, I. T	1
Dec. 29,	McCurtain, I. T	1
1905–Jan. 20,	Colgate, I. T	1
Feb. 14,	Williamstown, Pa	1
Feb. 17,	Shenandoah City, Pa	1
Feb. 20,	Virginia City, Ala	111
Feb. 25,	McAlester, I. T	2
Feb. 26,	Wilcoe, W. Va	7
March 5,	Burton No. 13 Mine, Iowa.	1
March 11,	Park Place, Pa	1
March 18,	Red Ash, W. Va	24
March 20,	Kingston, Pa	1
March 22,	Princeton, Ind	9
April 3,	Ziegler, Ill	53
April 3,	West End, Pa	1
April 20,	Mt. Pleasant, Pa	1
April 20,	Cabin Creek, W. Va	6
April 27,	Maple Hill, Pa	2
April 27,	Dubois, Pa	13
April 30,	Wilburton, I. T	13
May 8,	Seneca, Pa	1
May 15,	Duquoin, Ill	1
June 2,	Phœnix Mine, W. Va	1
June 19,	Hartshorne, I. T	1
June 26,	Harry E. Mine, Pa	1
July 5,	Vivian, W. Va	5
July 6,	No. 11 Shaft, Pa	1
July 12,	Buck, I. T	2
July 16,	Lutie, I. T	1
July 16,	Carlisle Mine, Fayette Co., W. Va	2
July 18,	Lance, Luzerne Co., Pa	1
July 20,	Victoria No. 2 Mine, W. Va.	1
July 24,	Mahanoy City, Pa	1
Aug. 3,	Midvale Slope, Luzerne Co., Pa	1
Aug. 19,	No. 6 Shaft, Luzerne Co., Pa.	1
Aug. 25,	Phœnix Park, Pa	1
Sept. 12,	Boston Run, Pa	1
Oct. 2,	Brookside, Pa	1
Oct. 5,	No. 14 Shaft, Pa	1
Oct. 15,	Johnsville, I. T	2
Oct. 19,	Silver Creek, Pa	1
Oct. 29,	Hazelkirk, Pa	5
Nov. 4,	Vivian, W. Va	7
Nov. 10,	Buck Run, Pa	1
Nov. 15,	Penn, Pa	1
Nov. 15,	Bentleyville, Pa	6
Nov. 16,	Hughes, I. T	1
Nov. 22,	Wilburton, I. T	2
Nov. 23,	Stevens, Pa	1
Dec. 1,	Diamondville, Wyo	18
Dec. 7,	Maxwell, Pa	1
Dec. 12,	Buck Mt., W. Va	1
Dec. 13,	Luke Fidler, Pa	4
Dec. 15,	Harry E. Mine, Pa	1
Dec. 21,	Mt. Lookout, Pa	1
1906–Jan. 2,	Wanamie No. 18, Pa	1
Jan. 4,	Coaldale, W. Va	22
Jan. 15,	No. 14 Tunnel, Luzerne Co., Pa	2
1906–Jan. 18,	Detroit, W. Va	18
Jan. 24,	Witteville, I. T	14
Jan. 29,	Sherman, Ill	2
Feb. 1,	Bald Knob, W. Va	3
Feb. 8,	Parral, W. Va	27
Feb. 19,	Maitland, Colo	14
Feb. 27,	Piper, Ala	12
March 3,	Midvale Slope, Luzerne Co., Pa	1
March 4,	Packer No. 3, Pa	1
March 21,	L. C. & N. Co. No. 10, Pa.	1
March 22,	Century, W. Va	23
April 10,	Penn No. 10, Cambria Co., Pa	1
April 22,	Cautro, Colo	19
May 7,	Storrs, Pa	1
May 14,	Parrish, Pa	2
June 9,	Alaska, Pa	1
June 11,	Nesquehoning No. 1, Pa	1
June 12,	Richards, Pa	1
June 18,	Piney No. 1, W. Va	1
June 19,	Avondale, Pa	1
June 21,	Jamison No. 3, Pa	1
July 2,	Keystone, W. Va	4
July 16,	Cameron, Pa	1
July 19,	Huger, W. Va	4
July 20,	Nottingham, Pa	3
Aug. 1,	Warrior Run, Pa	1
Aug. 6,	North Shaft No. 7, Pa	6
Aug. 21,	Vulcan, Pa	2
Aug. 23,	Sugar Notch, Pa	2
Sept. 6,	Johnson City, Ill	1
Oct. 2,	Muhlenburg Co., Ky	1
Oct. 3,	Pocahontas, Va	36
Oct. 5,	Blossburg, N. Mex	10
Oct. 12,	Exeter, Pa	1
Oct. 18,	Nottingham, Pa	1
Oct. 19,	Prospect Shaft, Luzerne Co., Pa	1
Oct. 24,	Johnstown, Pa	7
Oct. 25,	Vulcan, Pa	1
Oct. 27,	Herrin, Ill	2
Nov. 7,	Midvalley, Pa	1
Nov. 13,	Central, Pa	1
Nov. 21,	Central, Pa	2
Nov. 26,	Palmer, Wash	1
Dec. 6,	Woodward, Pa	1
Dec. 6,	Bettonwood, Pa	4
1907–Jan. 3,	Shenandoah City, Pa	1
Jan. 17,	Pine Ridge, Pa	1
Jan. 18,	Harrisburg, Ill	1
Jan. 23,	Primero, Colo	24
Jan. 26,	Penco, W. Va	12
Jan. 29,	Fayetteville, W. Va	85
Jan. 29,	Nottingham, Pa	1
Feb. 4,	Thomas No. 25, V. Wa	25
Feb. 12,	Lansford No. 4, Pa	1
March 2,	Holden, Pa	7
March 2,	Woodward, Pa	2
March 5,	Tunnel Ridge, Pa	1
March 16,	Tacoma, Va	11
March 16,	Howard, Pa	1

Mine Explosions—Continued

Date	Place	Number of Fatalities
1907–March 19,	Nottingham, Pa.	1
March 23,	Barnum No 2, Pa.	1
March 23,	Sugar Creek, Ohio.	3
April 6,	Pettibone, Pa.	1
April 8,	Locust Spring, Pa.	2
April 11,	Consolidation Coal Co., Iowa.	1
April 24,	Midway, Colo.	1
April 26,	Black Diamond, Wash.	7
May 1,	Whipple, W. Va.	18
May 15,	No. 4 Shaft, Luzerne Co., Pa.	1
June 17,	Reliance, Pa.	1
June 18,	Johnson No. 1, Pa.	7
July 17,	North Mahanoy, Pa.	1
July 20,	No. 14 Shaft, Pa.	2
Aug. 1,	Penn, Pa.	1
Aug. 20,	Storrs No. 3, Pa.	1
Aug. 23,	Franklin, Washington.	3
Aug. 29,	Stanton No. 7, Pa.	1
Sept. 21,	So. Wilkes-Barre, Pa.	4
Sept. 28,	Mineral Spring Shaft, Pa.	1
Sept. 30,	Storrs No. 1, Pa.	1
Sept. 7,	W. Frankfort, Ill.	4
Nov. 12,	Harry E. Mine, Pa.	1
Nov. 14,	So. Wilkes-Barre, Pa.	1
Nov. 27,	Stanton No. 7, Pa.	1
Dec. 1,	Naomi, Pa.	34
Dec. 6,	Monongah, W. Va.	358
Dec. 16,	Yolande, Ala.	16
Dec. 18,	Darr, Pa.	239
Dec. 20,	Pictou Mine, Colo.	1
Dec. 23,	Buck, Okla.	1
Dec. 31,	Carthage, New Mexico.	11
1908–Jan. 9,	Scott, Pa.	3
Jan. 11,	Harrisburg, Ill.	1
Jan. 13,	Conyngham, Pa.	1
Jan. 13,	Woodward, Pa.	2
Jan. 25,	Stonington, Ill.	1
Jan. 30,	Bachman, W. Va.	9
Jan. 31,	No. 14 Shaft, Pa.	1
Feb. 7,	Port Hood, N. S.	10
Feb. 10,	Baltimore No. 5, Pa.	2
Feb. 10,	South Carrollton, Ky.	9
Feb. 12,	Eldorado, Ill.	1
Feb. 15,	Chant, Okla.	2
Feb. 17,	Eagle Hill, Pa.	1
Feb. 24,	No. 14 Shaft, Pa.	1
Feb. 27,	Roseta, Mexico.	83
March 15,	Standard Mine, Colo.	1
March 24,	No. 6 Shaft, Luzerne Co., Pa.	1
March 28,	Hanna, Wyo.	59
March 31,	Oakdale Mine, Colo.	1
March 31,	Sugar Notch No. 9, Pa.	1
April 8,	Wm. Penn, Pa.	2
April 14,	Champion Mine, Colo.	1
April 18,	Woodward, Pa.	2
April 23,	Ellsworth No. 1, Pa.	1
May 11,	No. 7 Colliery, Luzerne Co., Pa.	4
May 12,	Mt. Lookout, Pa.	12
1908–May 23,	Otto, Pa.	2
May 27,	Salineville, Ohio.	3
June 19,	Ellsworth No. 1, Pa.	3
July 8,	Lance No. 11, Pa.	1
Aug. 12,	Agnes, Washington Co., Pa.	1
Aug. 28,	Packer No. 2, Pa.	2
Aug. 31,	Brewer, Okla.	1
Sept. 1,	Nonpareil Mine, Colo.	1
Sept. 3,	Shenandoah City, Pa.	2
Sept. 8,	Coaldale No. 1, Pa.	1
Sept. 17,	Sugar Notch No. 9, Pa.	1
Sept. 26,	Hartshorne, Okla.	1
Oct. 7,	Phœnix Park, Pa.	2
Oct. 8,	No. 14 Shaft, Luzerne Co., Pa.	1
Oct. 14,	Carbonado, Wash.	1
Oct. 22,	Packer No. 3, Pa.	1
Oct. 29,	Short Creek, Ala.	6
Oct. 31,	Wilburton, Okla.	1
Nov. 5,	Rend City, Ill.	4
Nov. 6,	Buck Mt., Pa.	1
Nov. 9,	DeRush Mine, Colo.	1
Nov. 9,	Oxford, Pa.	1
Nov. 10,	Maxwell No. 20, Pa.	1
Nov. 13,	Woodward, Pa.	1
Nov. 18,	Shenandoah City, Pa.	1
Nov. 19,	Rend City, Ill.	3
Nov. 23,	Black Diamond, Pa.	1
Nov. 28,	Marianna, Pa.	154
Dec. 29,	Lick Branch, W. Va.	54
1909–Jan. 10,	Ziegler, Ill.	26
Jan. 12,	Lick Branch, W. Va.	69
Jan. 19,	Chancellor, Colo.	6
Jan. 25,	Boswell, Pa.	5
Jan. 29,	Sewickley, Pa.	2
Jan. 29,	N. W. I. & S. Co., Wash.	2
Feb. 2,	Short Creek, Ala.	18
Feb. 9,	Ziegler, Ill.	3
Feb. 16,	West Frankfort, Ill.	4
March 2,	No. 14 Shaft, Luzerne Co., Pa.	8
March 20,	Evansville, Ind.	6
March 23,	Silver Creek, Pa.	1
March 29,	Coahuila, Mexico.	35
March 31,	Beury, W. Va.	6
April 7,	Coast Coal Co., Wash.	1
April 9,	Windber, Pa.	7
April 13,	Superior Mine, Ind.	20
April 15,	Air Shaft No. 3, Marion Co., W. Va.	3
May 1,	No. 5 Mine, Luzerne Co., Pa.	1
May 7,	No. 6 Mine, Luzerne Co., Pa.	1
May 22,	Gilberton, Pa.	1
June 2,	Richards, Pa.	1
June 3,	Richards, Pa.	1
June 3,	Lytle, Pa.	1
June 16,	No. 8 Colliery, Schuylkill Co., Pa.	1
June 18,	Spring Brook, Pa.	1
June 23,	Wehrum, Pa.	21

Mine Explosions—Continued

Date	Place	Number of Fatalities
1909–June 23,	Nottingham, Pa.	2
July 6,	Tollerville, Colo.	9
July 2,	Grayson, Ill.	2
July 15,	Newcastle, Pa.	1
July 19,	Revere, Pa.	1
Aug. 18,	Richards, Pa.	1
Aug. 30,	No. 8 Colliery, Schuylkill Co., Pa.	1
Sept. 10,	Nottingham, Pa.	1
Sept. 16,	Parral, W. Va.	1
Sept. 29,	W. Eureka No. 7, Pa.	1
Oct. 3,	Roslyn, Wash.	10
Oct. 5,	Nanaimo, B. C.	30
Oct. 11,	Sesser, Ill.	1
Oct. 21,	Hartshorne, Okla.	10
Oct. 31,	Franklin No. 2, Cambria Co., Pa.	13
Nov. 2,	Maple Hill, Pa	1
Nov. 5,	Prospect, Pa	1
Nov. 17,	Lance No. 11, Luzerne Co., Pa.	1
Nov. 17,	Reliance, Pa.	1
Nov. 21,	West Frankfort, Ill.	1
Nov. 29,	Johnson City, Ill.	1
Dec. 11,	Clay, Ky.	7
Dec. 23,	Herrin, Ill.	8
Dec. 27,	Centralia, Ill.	4
1910–Jan. 11,	Nottingham, Pa.	7
Jan. 17,	Ernest No. 2, Pa.	5
Jan. 20,	Woodward, Pa.	1
Jan. 31,	Primero, Colo.	75
Feb. 1,	Browder, Ky.	34
Feb. 2,	Palau Mine, Mexico	57
Feb. 3,	Oxford, Pa.	1
Feb. 5,	Ernest No. 2, Pa.	6
Feb. 8,	Stearns, Ky.	6
Feb. 21,	Lytle, Pa.	1
Feb. 22,	Pettibone, Pa.	2
Feb. 26,	Christopher, Ill.	1
March 2,	Parrish, Pa.	1
March 12,	So. Wilkes-Barre, Pa.	7
March 21,	Lansford, Pa.	1
March 24,	Allison, Pa.	2
March 26,	Cambria, Okla.	4
March 31,	Wilburton, Okla.	6
March 31,	Maxwell No. 20, Pa.	1
April 20,	Mulga, Ala.	40
April 21,	Amsterdam, Ohio.	15
May 5,	Palos, Ala.	83
May 28,	St. Nicholas, Pa.	1
June 15,	Eddy Creek, Pa.	2
June 15,	So. Wilkes-Barre No. 5, Pa.	1
June 23,	Woodward, Pa.	1
July 1,	Kaska William, Pa.	1
July 2,	Mt. Lookout, Pa.	1
July 6,	Wanamie, Pa.	1
July 11,	Ewen, Pa.	1
Sept. 9,	Ewen, Pa.	1
Sept. 30,	Palau No. 2 Mine, Menor, Mex.	78
Oct. 3,	Roslyn, Wash.	10
Oct. 8,	Starkville, Colo.	56
1910–Oct. 13,	Bliss, Luzerne Co., Pa.	2
Nov. 4,	Yolande, Ala.	5
Nov. 6,	Black Diamond, Wash.	16
Nov. 8,	Delagua, Colo.	79
Nov. 11,	Panama, Ill.	6
Nov. 25,	Providence, Ky.	10
Nov. 14,	Greeno, Va.	11
Nov. 14,	Middleton Mine, W. Va.	2
Nov. 26,	Grayson, Ill.	2
Nov. 28,	Antlers, Okla.	13
Dec. 9,	Bellevue Mine, B. C.	31
Dec. 13,	Ravensdale, Wash.	2
Dec. 18,	Harrisburg, Ill.	1
Dec. 19,	Consolidation No. 47, W. Va.	2
Dec. 23,	Herrin, Ill.	8
Dec. 27,	Centralia, Ill.	4
Dec. 29,	Wick Haven, Pa.	1
1911–Jan. 3,	Sydney No. 3 Mine, Nova Scotia.	8
Jan. 20,	Carbon Hill, Va.	5
Jan. 25,	Pittston, Pa.	6
Feb. 9,	Cokedale Mine, Colo.	17
March 18,	West Mineral, Kan.	5
April 8,	Banner Mine, Ala.	128
April 14,	Herrin, Ill.	1
April 24,	Elk Garden, W. Va.	23
May 27,	Cameron Mine, Pa.	5
July 15,	Sykesville, Pa.	21
Aug. 1,	Standard Mine, W. Va.	4
Oct. 23,	Harrisburg, Ill.	9
Nov. 9,	Adrian Mine, Pa.	8
Nov. 18,	Bottom Creek Mine, W. Va.	18
Dec. 9,	Briceville, Tenn.	84
1912–Jan. 9,	Plymouth, Pa.	6
Jan. 13,	Windsor, Mo.	2
Jan. 15,	Regal Mine, Iowa	2
Jan. 17,	Central City, Ky.	5
Jan. 20,	Susie, Wyo.	5
March 7,	Merritt, B. C.	7
March 10,	Johnson City, Ill.	1
March 20,	McCurtain, Okla.	74
March 26,	Jed, W. Va.	83
April 15,	Tacoma, Wash.	1
April 21,	Madisonville, Ky.	5
April 23,	Panama, Ill.	2
June 18,	Hastings, Colo.	12
July 11,	Moundsville, W. Va.	8
July 16,	Carbon Hill, Va.	8
July 17,	So. Wilkes-Barre, Pa.	4
July 24,	D. & H. No. 2, Pa.	2
Aug. 13,	Abernant, Ala.	18
Aug. 30,	Piedmont, Colo.	2
Sept. 16,	Coral, Pa.	1
Oct. 1,	South Canon, Colo.	2
Oct. —,	Oak Hill Mine, Ind.	2
Oct. 25,	Mayer Coal Co., Iowa	2
Nov. 8,	Simpson, Colo.	1
Nov. 16,	Clifton Mine, Ill.	1
Nov. 22,	Peoria, W. Va.	2
Dec. 20,	Taylor Mine, Pa.	2

TABLE II.—*Explosions of Gas and Dust in Coal Mines of North America, by Months, 1839 to 1912, Inc.*

Month	Explosions Having Less than Five Fatalities				Explosions Having More than Five Fatalities				Total Explosions			
	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions
January	69	109	1.58	10.53	33	953	28.88	14.23	102	1,062	10.41	11.50
February	44	68	1.50	6.72	28	800	28.57	12.07	72	868	12.06	8.12
March	57	83	1.46	8.70	33	781	23.67	14.23	90	864	9.60	10.14
April	44	64	1.45	6.72	22	428	19.45	9.48	66	492	7.45	7.44
May	47	78	1.67	7.18	18	833	46.28	7.76	65	911	14.01	7.33
June	54	74	1.37	8.24	11	271	24.64	4.74	65	345	5.31	7.33
July	51	79	1.55	7.79	10	205	20.50	4.31	61	284	4.66	6.88
August	47	64	1.36	7.18	5	49	9.80	2.15	52	113	2.17	5.86
September	50	70	1.40	7.63	8	140	17.50	3.45	58	210	3.62	6.54
October	65	96	1.48	9.92	21	283	13.48	9.06	86	379	4.41	9.69
November	63	87	1.38	9.62	24	496	20.67	10.34	87	583	6.70	9.81
December	64	105	1.64	9.77	19	1,011	53.21	8.18	83	1,116	13.45	9.36
Totals	655	977	.....	100.00	232	6,250	.....	100.00	887	7,227	.....	100.00
Averages	.....	.....	1.49	.....	.....	.....	26.94	.....	.....	.....	8.15	.....

TABLE III.—*Production in Short Tons and Fatal Accidents Caused by Explosions of Gas and Dust in Coal Mines of United States*

Year	Production	Accidents	Fatalities	Accidents per Million Tons	Fatalities per Million Tons	Accidents in which 5 or more Fatalities Occurred: Number	Fatalities	Accidents per Million Tons	Fatalities Per Million Tons	Mines	Average Production Per Mine
1820	3,450										
1821	1,322										
1822	58 583										
1823	68,563										
1824	80,725										
1825	117,988										
1826	147,914										
1827	172,151										
1828	195,908										
1829	240,086										
1830	320,072										
1831	337,942										
1832	594,050										
1833	734,657										
1834	600,515										
1835	824,854										
1836	984,832										
1837	1,253,651										
1838	1,355,527										
1839	1,560,360	1	40	0.64	25.6	1	40	0.64	25.6		
1840	2,070,039										
1841	2,291,141										
1842	2,610,057										
1843	3,060,874										
1844	3,681,252										
1845	4,309,904										
1846	4,865,522										
1847	5,286,067										
1848	5,773,974										
1849	6,448,831										
1850	7,018,181									510	12,539
1851	8,734,525										
1852	9,816,664										
1853	10,570,288										
1854	11,977,102	1	19	0.08	1.51	1	19	0.08	1.51		
1855	12,926,673	1	55	0.08	4.27	1	55	0.08	4.27		
1856	13,546,925										
1857	13,340,189										
1858	13,974,478										
1859	15,633,175										
1860	14,610,042									622	23,045
1861	16,488,012										
1862	17,485,835										
1863	21,319,062										
1864	23,605,123										
1865	23,792,173										
1866	29,003,583										
1867	30,724,422										
1868	32,861,960										
1869	32,904,360	2	2	0.06	0.06						
1870	33,035,580	6	6	0.18	0.18					1,566	20,986
1871	46,885,080	13	21	0.28	0.45	1	5	0.02	0.10		
1872	51,453,399	11	21	0.21	0.41	1	7	0.02	0.14		
1873	57,602,480	10	20	0.17	0.35	1	10	0.02	0.18		
1874	52,605,920	14	18	0.26	0.34						
1875	52,348,320	6	8	0.12	0.15						
1876	55.280.000	16	31	0.30	0.58	2	14	0.04	0.26		

TABLE III.— *Continued.*

Year	Production	Accidents	Fatalities	Accidents per Million Tons	Fatalities per Million Tons	Accidents in which 5 or more Fatalities Occurred				Mines	Average Production per Mine
						Number	Fatalities	Accidents per Million Tons	Fatalities per Million Tons		
1877	60,501,760	2	9	0.03	0.15	1	7	0.02	0.12	.....	.....
1878	57,955,600	6	18	0.10	0.31	2	13	0.04	0.22	.....	.....
1879	68,105,799	7	17	0.10	0.25	2	11	0.03	0.16	.....	.....
1880	71,481,570	4	8	0.06	0.11	1	5	0.01	0.07	3,294	21,701
1881	85,881,030	8	56	0.09	0.65	3	50	0.04	0.58	.....	.....
1882	103,551,189	10	49	0.10	0.47	2	37	0.02	0.36	.....	.....
1883	115,707,525	16	36	0.15	0.31	1	10	0.01	0.09	.....	.....
1884	120,155,551	8	220	0.07	1.83	5	213	0.04	1.77	.....	.....
1885	111,160,295	4	17	0.04	0.16	2	12	0.02	0.10	.....	.....
1886	113,680,427	6	77	0.05	0.68	5	76	0.04	0.67	.....	.....
1887	130,650,511	4	23	0.03	0.18	1	18	0.01	0.14	.....	.....
1888	148,659,657	6	90	0.04	0.61	3	83	0.02	0.56	.....	.....
1889	141,229,513	9	18	0.06	0.13	.....	.....	.....	.....	2,583	53,578
1890	157,770,963	8	54	0.05	0.34	4	49	0.03	0.31	.....	.....
1891	168,566,669	19	156	0.12	0.93	3	128	0.02	0.76	.....	.....
1892	179,329,071	19	164	0.11	0.91	3	127	0.02	0.71	.....	.....
1893	182,352,774	12	67	0.07	0.37	7	60	0.04	0.33	.....	.....
1894	170,741,526	23	40	0.14	0.23	1	8	0.01	0.05	.....	.....
1895	193,117,530	18	200	0.09	1.04	8	188	0.04	0.97	.....	.....
1896	191,986,357	15	95	0.08	0.50	4	77	0.02	0.40	.....	.....
1897	200,229,199	27	60	0.14	0.30	3	22	0.02	0.11	.....	.....
1898	219,976,267	33	58	0.16	0.26	2	14	0.01	0.07	.....	.....
1899	253,741,192	26	118	0.10	0.47	7	96	0.03	0.38	.....	.....
1900	269,684,027	32	309	0.12	1.11	4	268	0.02	1.00	.....	.....
1901	293,299,816	30	110	0.10	0.38	6	67	0.02	0.23	.....	.....
1902	301,590,439	23	416	0.08	1.38	10	391	0.03	1.30	5,986	50,383
1903	357,356,416	41	257	0.12	0.72	6	207	0.02	0.57	.....	.....
1904	351,816,398	37	268	0.11	0.76	4	219	0.01	0.60	.....	.....
1905	392,722,635	53	326	0.13	0.83	13	277	0.03	0.70	.....	.....
1906	414,157,278	48	265	0.12	0.64	13	213	0.03	0.51	.....	.....
1907	480,363,424	46	899	0.10	1.87	14	854	0.03	1.78	.....	.....
1908	415,842,698	52	365	0.13	0.88	6	297	0.02	0.72	.....	.....
1909	460,814,616	49	295	0.11	0.64	17	249	0.04	0.54	6,079	75,804
1910	501,596,378	48	539	0.10	1.07	21	498	0.04	0.99	6,126	81,553
1911	490,601,317	14	334	0.03	0.63	12	329	0.02	0.67	6,179	79,398
1912	534,446,580	26	253	0.05	0.47	10	224	0.02	0.42	6,022	88,752

TABLE IV.—*Mine Explosions in Great Britain*

Date	Place	Number of Fatalities
1803–Sept. 25,	Wall's End	13
1805–Oct. 21,	Hebburn	35
Nov. 29,	Oxclose	38
1806–Nov. 28,	Killingworth	10
1808–Nov. 29,	Harraton	4
1809–Sept. 14,	Killingworth	12
1812–May 25,	Felling	92
Oct. 10,	Barrington Mill Pit	24
1813–July 17,	Collingwood Main	8
Sept. 28,	Hall Pit, Fatfield	32
Dec. 24,	Felling	22
1814–April 5,	Hawden Pit, Percy, Main	4
Aug. 12,	Hebburn	11
Sept. 9,	Leefield	4
1815–June 2,	Sheriff Hill	11
Dec. 18,	Townley	1
1817–June 30,	Row Pit, Harraton	38
1817–Sept. 25,	Jarrow	6
Aug. 5,	Wall's End	4
Nov. 3,	Ouston	1
Dec. 18,	Plain Pit, Rainton	27
1819–July 19,	Sheriff Hill	35
Oct. 9,	George Pit, Lumley	13
1821–Oct. 19,	Nesham's Newbottle	6
Oct. 23,	Wall's End	52
Oct. 23,	Felling	6
1823–Feb. 21,	Ouston	4
Nov. 3,	Plain Pit, Rainton	59
1824–Oct. 5,	George Pit, Lumley	14
Nov. 19,	Dolly Pit, Newbottle	11
1825–July 3,	Julit Pit, Fatfield	11
Oct. 5,	Hebburn	4
1826–Jan. 17,	Jarrow	34
May 30,	Townley	38

Mine Explosions in Great Britain—Continued

Date	Place	Number of Fatalities
1826–Sept. 5,	Heworth	5
Oct. 27,	Benwell	2
1828–March 15,	Jarrow	8
Sept. 1,	New Pit, Houghton	7
Nov. 20,	J. Pit, Washington	14
1829–Dec. 3,	Willington	4
1830–Aug. 3,	Jarrow	42
1831–July 9,	King Pit, Wreckington	3
Sept. 20,	Willington	7
1833–May 9,	Springwell	47
May 24,	Great Lumley	2
Nov. 1,	Black Fell	3
1835–June 18,	Wall's End	102
Nov. 19,	Burdon Main	11
1836–Jan. 28,	Down's Pit, Hetton	22
1837–Dec. 6,	Springwell	30
1838–Dec. 19,	Wall's End	11
1839–June 28,	St. Hilda	51
1840–June 6,	Haswell	1
1841–April 19,	Bigge Pit, Willington	32
Aug. 5,	Thornley	9
Aug. 17,	Haswell	1
1842–March 2,	West Cramlington	1
1843–April 5,	King Pit, Wreckington	28
1844–Jan. 18,	West Moor	5
Sept. 28,	Haswell	95
Oct. 15,	Coxlodge	1
Dec. 1,	Seghill	2
1849–June 10,	Ferndale, Eng	—
1851–March 15,	Victoria Pit	61
May 26,	Washington Pit, Newcastle	2
Aug. 19,	Washington Pit, Newcastle	35
Oct. 23,	Killingworth Pit, Newcastle	1
Oct. 31,	Kilingworth Pit, Newcastle	9
Dec. 20,	Warren Vale, Yorkshire	52
Dec. 22,	Ince Hall, Lancashire	13
1852–April 24,	Norley Hall, Lancashire	12
May 6,	Hebburn, Durham	22
May 10,	Middle Duffryn	65
May 20,	Coppull, Lancashire	36
Dec. 22,	Elsecar, Yorkshire	10
1853–March 12,	Risca, Monmouth	10
March 24,	Arley Mine	58
April 26,	Old Park, Worcestershire	11
July 1,	Bent Grange, Lancashire	20
1854–Feb. 18,	Ince Hall	89
1856–July 3,	Old Coal Pit, Monmouth	11
July 15,	Cymmer, Wales	114
Aug. 13,	Ramrod Hall, Staffordshire	11
May 24,	Cyme Avon, Glamorgan	12
1857–Feb. 19,	Lund Hill	189
May 27,	Tyr Nicholas, Monmouth	13
July 31,	Keys, Lancashire	40
1858–Feb. 2,	Bardsley	55
Feb. 25,	Lower Duffryn	19
May 28,	Bryndu, Glamorgan	12
Dec. 11,	Tyldesley, Lancashire	25
1860–Feb. 15,	Higham, Yorkshire	13
March 2,	Burradon	76
Aug. 3,	Winstanley, Lancashire	13
Nov. 6,	Lower Duffryn, Glamorgan	12
Dec. 1,	Risca, Wales	142
Dec. 20,	Hetton Sutton, Durham	22
1861–March 8,	Blaengwawr, Glamorgan	13
1861–Sept. 26,	South Mostyn, Flintshire	10
Nov. 1,	Shevington, Lancashire	13
1862–Feb. 19,	Gethin, Glamorgan	47
Nov. 22,	Walker, Northumberland	16
Dec. 8,	Edmunds Main, Yorkshire	59
1863–March 6,	Coxlodge, Northumberland	26
Oct. 17,	Morfa	39
Dec. 9,	Wynnstay, Denbigh	13
Dec. 26,	Lleynoi, Glamorgan	15
1865–June 16,	Bedwelty	26
Dec. 20,	Cethin Cyforthfa, Glamorgan	34
1866–Jan. 23,	High Brooks, Lancashire	30
May 4,	Garswood Park, Lancashire	12
June 14,	Victoria Pit	38
Oct. 31,	Palton	24
Dec. 12,	Oaks Colliery	361
Dec. 13,	Talk O'The Hill	91
1867–Aug. 20,	Garswood	14
Nov. 8,	Ferndale, Wales	178
Nov. 11,	Homer Hill, Staffordshire	12
1868–Sept. 30,	Wynnstay, Denbigh	10
Nov. 25,	Hindley Green	62
Dec. 26,	Haydock, Lancashire	26
1869–April 1,	Highbrooks	37
June 10,	Ferndale, Glamorgan	53
July 21,	Haydock	59
Oct. 22,	Newbury, Somerset	11
Nov. 15,	Low Hall, Lancashire	27
1870–Feb. 14,	Morfa, Glamorgan	30
July 7,	Silverdale, Staffordshire	19
July 23,	Charles Colliery	19
Aug. 19,	Brynn Hall	20
1871–Jan. 10,	Renishaw	20
Feb. 24,	Pentre	38
March 2,	Victoria, Monmouth	19
Sept. 6,	Moss Pit	70
Oct. 25,	Seaham	26
1872–Feb. 14,	Maesteg Marthyr, Glamorgan	11
March 28,	Lover's Lane, Lancashire	27
Oct. 7,	Morley	34
1873–Feb. 18,	Talke, Staffordshire	18
1874–April 14,	Dunkinfield	54
April 18,	Ince Hall, Lancashire	15
Nov. 20,	Rawmarsh, Yorkshire	23
Dec. 24,	Bignall Hill, Staffordshire	17
1875–April 30,	Bunker's Hill	43
Dec. 4,	Tredegar, Monmouth	23
Dec. 6,	Llan, Glamorgan	16
Dec. 6,	Swaithe Main	143
Dec. 15,	Mons Colliery	—
1876–Dec. 18,	South Wales, Monmouth	23
1877–Jan. 23,	Stonehill, Lancashire	18
Feb. 7,	Foggs, Lancashire	10
March 10,	Wiegfach, Glamorgan	18
Oct. 11,	Pemberton, Lancashire	36
Oct. 22,	Blantyre	207
1878–March 8,	Barwood Colliery	17
March 12,	Unity Brook	43
March 27,	Apedale, Staffordshire	23
June 7,	Haywood Wood	189
Sept. 11,	Aberdarne, Monmouth	268
1879–March 4,	Victoria, Yorkshire	21

Mine Explosions in Great Britain—Continued

Date	Place	Number of Fatalities
1879–Jan. 13,	Dinas	63
July 2,	Blantyre	28
1880–Jan. 21,	Leycett Colliery	62
July 15,	Risca, Wales	120
Sept. 8,	Seaham	164
Dec. 10,	Naval, Wales	101
1881–Feb. 7,	Whitfield	25
Dec. 19,	Abram	48
1882–Feb. 16,	Trimdon Grange	74
April 18,	Tudhoe	37
April 19,	West Stanley, Durham	13
April 25,	Whitehaven, Cumberland	4
May 2,	Baddesley	23
Nov. 7,	Claycross No. 7, Parkhouse, Derbyshire	45
1883–Oct. 18,	Wharncliffe	20
Nov. 7,	Altham	68
1884–Jan. 27,	Penygraig, Glamorgan	14
Sept. 6,	Hall End, Staffordshire	7
Nov. 8,	Pochin, Monmouth	14
1885–March 2,	Usworth	42
April 8,	Great Fenton, Staffordshire	8
June 18,	Clifton Hall	178
Aug. 20,	Apedale, Staffordshire	9
Dec. 23,	Mardy	81
1886–Aug. 13,	Woodends Pit	38
Sept. 10,	Dean Lane, Somerset	10
Oct. 2,	Altofts, Yorkshire	22
Dec. 2,	Elemore, Durham	28
1887–Feb. 18,	Ynshir Colliery	39
May 28,	Udston Colliery	73
1888–April 19,	St. Helens, Cumberland	30
1889–Jan. 18,	Hyde	23
Jan. 28,	Dean Lane	4
Jan. 31,	Kirkland, Scotland	1
Feb. 25,	Lower Llanmorlais, Wales	2
March 13,	Brynmally	20
March 21,	Dipton	1
April 19,	St. Helens	30
June 27,	Dairy Farm	1
July 9,	Cwmbran, Wales	1
July 23,	Bothwell Park No. 2, Scotland	1
Aug. 17,	Chell	3
Aug. 21,	Whitwood	2
Oct. 12,	Lammark No. 2, Scotland	1
Oct. 16,	Mossfields	64
Oct. 25,	Aslmore Park	1
Oct. 31,	Saltwells	2
Nov. 4,	Hebburn A	6
Nov. 4,	Meiklehill No. 5, Scotland	2
Dec. 26,	Barrwood No. 1, Scotland	1
1890–Jan. 22,	Glyn	5
Feb. 6,	Llanercy, Wales	176
March 3,	Devon, Scotland	1
March 10,	Morfa, Wales	87
March 24,	Gilvertheld No. 2, Scotland	1
April 4,	Hallside, Scotland	1
April 6,	Broomhouse, Scotland	1
May 2,	Thorncliffe	2
May 16,	Aberdure No. 9, Wales	2
May 16,	Vochriw, Wales	1
May 19,	Shetton Deep Pit	1
June 10,	Holmes, Scotland	1
Aug. 2,	Braich Y Gimmer, Wales	1
1890–Aug. 28,	Malago Vale Argus Pit, Wales	1
Aug. 30,	Kinneil, Scotland	2
Sept. 8,	Butterworth Hall	1
Sept. 14,	Hanby Northwood Pit	1
Sept. 19,	Nether Croy No. 1, Scotland	1
Oct. 13,	Eastfield, Scotland	1
Nov. 1,	Peel Hall White	1
Nov. 3,	Raverhead	1
Dec. 19,	Longrigg, Scotland	1
1891–Jan. 7,	Dowlais South Tunnel Pit, Wales	1
Feb. 10,	Beamish Second Pit	2
March 6,	Oak Farm	1
March 24,	Dearham	1
April 2,	Apedale Sladderhill Pit	10
May 9,	Shelton No. 1	1
May 13,	Franch, Wales	1
May 13,	Brithdir, Wales	1
May 14,	Ashton Moss	2
June 19,	Llanbradach, Wales	1
July 16,	Llantwit Red Ash, Wales	1
July 23,	Addiewell, Scotland	1
Aug. 1,	Ross, Scotland	2
Aug. 2,	Haughhead, Scotland	1
Aug. 7,	Glass Houghton, Eng	2
Aug. 23,	Abarcanaid, Wales	2
Aug. 25,	Rhos Llantwit, Wales	2
Aug. 31,	Malago Vale	10
Sept. 10,	Low Hall	2
Oct. 1,	Aberdare Merthyr, Wales	1
Oct. 13,	Backworth C Pit	1
Nov. 11,	Meiros, Wales	3
Dec. 9,	Montagu Main	2
1892–Feb. 8,	Dinnington	1
Feb. 10,	Hall End	1
Feb. 19,	Gadis Graig Pit, Wales	1
May 13,	Coxlodge	1
May 19,	Car House	1
June 1,	Browneyside, Scotland	1
June 17,	Holmes, Scotland	1
July 28,	Gartshore No. 3, Scotland	1
Aug. 8,	Ebbw Vale Marine, Wales	1
Aug. 26,	Park Slip, Wales	112
Sept. 24,	Netherton Old	1
Oct. 11,	Gartshore No. 9, Scotland	1
1893–Feb. 11,	Whitwick No. 6	2
Feb. 11,	Werfa, Wales	1
April 21,	Himley	2
May 1,	Sharlston	3
May 5,	Fishley	1
May 24,	Dunston	2
June 30,	Herbertshire No. 3, Scotland	2
June 30,	Butterknowle	1
July 4,	Combo, Thornhill	139
Aug. 19,	Hattonrig No. 3, Scotland	1
Oct. 13,	Camerton	2
Oct. 24,	Elms	1
Nov. 13,	Cowdenbeath, Scotland	1
Dec. 2,	Hem Heath	1
Dec. 8,	Wooley	1
1894–Jan. 1,	Gorseinon, Wales	1
March 23,	Ayr-Sundrum No. 3, Scotland	1
April 18,	Portland No. 5, Scotland	1

Mine Explosions in Great Britain—Continued

Date	Place	Number of Fatalities
1894–May 19,	Allerton Main	3
June 23,	Albion, Wales	290
Aug. 1,	Llwynpia No. 3, Wales	1
Aug. 8,	Bank House	1
Aug. 9,	Rowley Hall	1
Aug. 17,	Black Boy	1
Sept. 6,	West Auckland	1
Sept. 10,	Radcliffe	1
Sept. 21,	East Connock	1
Sept. 26,	Wednesbury	1
Sept. 27,	International Anthracite, Wales	1
Oct. 15,	Common No. 10, Scotland.	1
Oct. 16,	Harecastle & Woodshutts Col.	1
Oct. 17	Ellismuer's No. 2, Scotland	1
Oct. 25,	Blair Hall, Scotland	1
Oct. 27,	Common No. 14, Scotland.	1
Oct. 27,	Sandwell Park	4
Nov. 10,	Gadlys-Dyllas, Wales	2
Nov. 22,	Ashmore Park	1
1895–Feb. 6,	Timsbury	2
Feb. 9,	Morlais Vale, Wales	2
Feb. 13,	Glynea, Wales	1
Feb. 25,	Bwllfa Dare, Wales	1
March 15,	Malago Vale	2
April 6,	Seafield, Scotland	1
April 22,	Heddon	1
June 23,	Quarter, Scotland	13
May 6,	Llantwit Red Ash, Wales	1
June 18,	Bullfield	2
July 10,	Crawshays Castle, Wales	3
July 16,	Abercrave, Wales	1
Aug. 1,	Aberdare Merthyr, Wales	1
Aug. 1,	Aldridge	2
Aug. 14,	Pentland, Scotland	1
Sept. 14,	Lea Green	1
Oct. 1,	Shakerley	5
Nov. 5,	Tannockside No. 2, Scotland	1
Oct. 11,	Blackwell A	7
Nov. 21,	Newmarket	2
1896–Jan. 11,	West Cannock	1
Jan 11,	Llanmorlais, Wales	2
Jan. 27,	Ferndale Nos. 7 & 8, Wales	57
Jan. 28,	Blaennant, Wales	1
Feb. 1,	Foxhole, Wales	1
Feb. 4,	Saltwells,	1
Feb. 10,	Axwell Park	2
Feb 29,	Lammark No. 2, Scotland.	1
March 28,	Meiros, Wales	3
March 30,	Straiton, Scotland	1
April 13,	Brancepeth	20
April 27,	Swalwell	1
April 30,	Micklefield	63
May 6,	Birkrigg, Scotland	1
June 8,	Carfin No. 6, Scotland	1
June 15,	Bellfield, Scotland	1
June 19,	Abercave, Wales	1
Aug. 4,	Main Bryncoch, Wales	7
Aug. 17,	Sough Duffryn, Wales	1
Aug. 17,	Gawn	1
Sept. 16,	Cornsilloch, Scotland	2
Sept. 26,	Primrose Main	1
Dec. 3,	Drumsmudden, Scotland	1
Dec. 5,	Seaton Delaval	1
1896–Dec. 29,	Garscube, Scotland	1
1897–Jan. 5,	Broadeak, Wales	5
March 12,	Bothwell Park, Scotland	1
March 18,	Knowle	1
April 1,	Gadlys, Wales	1
May 13,	Darmgavd, Scotland	2
July 3,	Dalmeny, Scotland	1
Aug. 6,	Raveresthrope	3
Sept. 29,	Drumpeller No. 3 and No. 4, Scotland	1
Oct. 11,	Newliston, Scotland	1
Nov. 29,	Olvecote Tamworth	1
Dec. 10,	East Cannock	1
Dec. 20,	Blaendare	1
1898–Jan. 28,	Drumpeller Nos. 3 and 4, Scotland	4
March 5,	Whinney Moor	1
March 31,	Kinneal, Scotland	3
Aug. 3,	Millfield	1
Aug. 17,	Rosehall, Scotland	1
Aug. 21,	St. Helens	2
Sept. 6,	Greasbro	2
Sept. 9,	Ayr, Scotland	7
Sept. 9,	Thankerton, Scotland	1
Sept. 29,	Annagher No. 5, Ireland	3
Oct. 23,	Nethercroy, Scotland	1
Nov. 3,	South Elswick	1
1899–Jan. 9,	Gilbertfield No. 2, Scotland.	1
Jan. 10,	Pumpherston, Scotland	1
Feb. 17,	Walbottle	1
March 11,	Cadeby Main	2
March 18,	Gertshore No. 9, Scotland.	1
April 4,	Drumpeller Nos. 3 and 4, Scotland	1
May 15,	Etherty Grange	1
May 22,	Benarty, Scotland	2
May 24,	Woodhall No. 1, Scotland.	1
May 25,	Millfield	4
June 9,	Ynisgeinon, Wales	1
June 19,	Brownhill	1
June 20,	Holytown No. 5, Scotland.	2
Aug. 15,	Brandon	6
Aug. 15,	Wester Gartshore, Scotland.	1
Aug. 18,	Llest, Wales	19
Oct. 10,	Hamilton Palace No. 1, Scotland	1
Oct. 26,	Colderbank No. 1, Scotland	1
Nov. 3,	Bedlington A Pit	1
Nov. 23,	Point of Air	2
Dec. 6,	Kinneil, Scotland	2
Dec. 25,	Longriggend, Scotland	1
Dec. 27,	Ingliscard, Scotland	1
1900–Jan. 8,	Middrie, Scotland	1
Feb. 12,	Carbarns, Scotland	3
Feb. 22,	Glespin, Scotland	1
March 8,	Hinley	1
March 12,	Pumpherston, Scotland	1
March 25,	Drumbrook No. 2, Scotland	1
March 26,	Cowdenbeath, Scotland	1
April 7,	Cadley Hill	2
April 9,	Walsall Wood	1
April 12,	Holytown No. 1, Scotland.	2
May 28,	Clynhebog, Wales	2
June 10,	Clynmil Drift, Wales	1
June 22,	Garw Feehan, Wales	1

Mine Explosions in Great Britain—Continued

Date	Place	Number of Fatalities
1900–Aug. 12,	Tannochside No. 3, Scotland	1
Aug. 17,	Portland No. 5, Scotland	6
Sept. 3,	Kenmuirhill No. 2, Scotland	1
Oct. 3,	Hattonrigg Nos. 3 and 4, Scotland	1
Oct. 9,	Fieldhouse	1
Nov. 16,	Preston	4
Nov. 24,	Gilmithscroft No. 4, Scotland	1
Nov. 30,	Deelmont No. 1, Scotland	1
Dec. 3,	Oatlands	1
Dec. 9,	Hamstead	1
Dec. 28,	Ross, Scotland	1
1901–Jan. 16,	Ystradgynlas	1
Jan. 23,	Kirkwood, Scotland	1
Jan. 25,	Haughhead, Scotland	2
Jan. 27,	Holytown No. 8, Scotland	1
Feb. 28,	Blaendare Slope	2
March 25,	Dinas, Wales	4
April 1,	Sandback	2
April 10,	Orrell	4
May 6,	Tursdale	1
May 16,	Gilmithscroft No. 3, Scotland	1
May 24,	Universal, Wales	81
May 27,	Talk O'The Hill	4
June 26,	Meiros, Wales	1
July 8,	Stonedough	4
July 20,	Fochriw, Wales	1
Aug. 19,	Hamstead	2
Aug. 26,	New Dale	1
Sept. 10,	Llanbradach, Wales	8
Sept. 18,	Pencoed, Wales	1
Nov. 12,	Edmondsley	1
Dec. 8,	Cornsilloch No. 3, Scotland	2
1902–March 3,	Pemberton	2
April 2,	Garwood Hall No. 9	9
April 3,	Glencraig, Scotland	4
April 19,	Bernarty, Scotland	1
May 14,	Todmorten Moor	1
May 21,	Dunnington	1
June 4,	Folchriw, Wales	8
July 4,	Woodhorn	3
July 8	Elba, Wales	1
July 9,	Wrexham & Action	1
July 19,	Collena, Wales	1
July 26,	Shut End No. 13	1
July 26,	Cleland, Scotland	1
July 29,	Darran, Wales	1
Aug. 1,	Pumpherston, Scotland	3
Aug. 15,	Skellyton No. 3, Scotland	2
Aug. 23,	Park Hill	1
Sept. 3,	McLaren, Wales	16
Oct. 3,	Blair No. 2, Scotland	1
Oct. 15,	Holmside	1
Nov. 25,	Prior Lee No. 2	1
Dec. 1,	East Plean No. 3, Scotland	1
Dec. 9,	Orrell	2
1903–Feb. 9,	Stanrigg, Scotland	1
Feb. 24,	Aldridge No. 1	1
Feb. 25,	Darran, Wales	2
Mar. 23,	Ferndale No. 5, Wales	1
1903–June 10,	Biggarford, Scotland	1
July 17,	Bogleshole, No. 4, Scotland	2
July 24,	Garriongill, Scotland	1
Oct. 5,	Westburg No. 2, Scotland	1
Nov. 9,	Llwynypia No. 3, Wales	1
Nov. 12,	Stargate Pit,	1
Dec. 11,	Braidhurst, Scotland	2
1904–Jan. 3,	Saltwells No. 27	1
Jan. 7,	St. Helens	2
Jan. 19,	Calder	1
March 19,	Saltwells No. 27	1
May 16,	Shirebrook	2
June 13,	Watergate	1
June 14,	Allanshaw No. 1, Scotland	2
June 25,	St. Johns	1
July 25,	Drumbow, Scotland	1
Aug. 23,	Backworth	1
Aug. 23,	Sneyd	3
Sept. 1,	Blantyre Ferme, Scotland	1
Sept. 6,	Broomhouse, Scotland	1
Sept. 26,	Rosehall No. 14, Scotland	1
Oct. 2,	Swinhill No. 2, Scotland	1
Dec. 31,	Bowhill, Scotland	1
1905–Jan. 21,	Elba, Wales	11
March 10,	Cambrian Coll., Wales	33
July 11,	Nat. Coll., Wales	119
1906–June 1,	Count Herbert, Glamorgan	5
Nov. 10,	Albion, Glamorgan	6
Dec. 17,	Urpeth Coll	4
Dec. 14,	Wingate Grange	25
1907–Nov. 26,	Whitehaven	5
March 19,	Benwell	4
Dec. 14,	Dinas Main, Wales	7
March 5,	Genwen, Wales	6
1908–Jan. 28,	Pantyffynnon Coll., Wales	2
Feb. 20,	Washington Glebe	14
Feb. 26,	Tibshelf No. 1, Derbyshire	1
Apr. 19,	Norton Hill	10
Aug. 18,	Maypole	75
Dec. 11,	Blaenavon Coll	3
1909–Jan. 19,	Ackton Hall Coll	2
Feb. 1	Littleburn Coll	1
Feb. 2,	Calderbank No. 2, Scotland	1
Feb. 17,	West Stanley	168
June 27,	Ferniegair No. 1, Scotland	3
Aug. 23,	Old Roundwood	1
Oct. 1,	Graig Merthyr, Wales	5
Oct. 20,	Berryhill No. 2	4
Oct. 22,	Caedean, Wales	4
Oct. 29,	Darran, Wales	27
Nov. 8,	Polquhairn, Scotland	1
Dec. 17,	Gurnos Coll., Wales	2
1910–Jan. 19,	Jammage Coll	1
May 11,	Wellington Mine, Whitehaven	136
July 21,	Rotherham Main Coll	3
Aug. 5,	Main Coal Colliery	2
Aug. 13,	Llanmorlais Colliery, Wales	1
Dec. 21,	Pretoria Colliery, Bolton	344
1911–April 12,	Ynyscedwyn Coll., So. Wales	1
April 30,	North Motherwell Coll., Scotland	1
June 8,	Seven Sisters, So. Wales	1

Mine Explosions in Great Britain—Continued

Date	Place	Number of Fatalities
Aug. 19,	Meiklehill Coll., Scotland..	1
Aug. 21,	Maltby Main Coll., Doncaster.....	3
Sept. 15,	Netherton Coll., Worcestershire.....	3
Sept. 16,	Cwmaman Coll., So. Wales	2
Oct. 15,	Dechmont No. 3, Scotland.....	1
Oct. 23,	Killan Coll., So. Wales.....	1
Oct. 26,	Annandale Coll., Scotland.	1
Nov. 3,	Holytown Coll., Scotland..	1
Nov. 25,	Jamage Coll., No. Staffordshire.....	6
1912–Jan. 6,	Hunwick Coll., Durham...	1
Feb. 2,	Bentley Coll., Doncaster...	3
Feb. 4,	Newbottle Coll., Scotland..	1
Feb. 24,	Norton Coll., No. Staffordshire.....	1
Feb. 26,	Ladyshore Coll.,.....	1
March —,	Great Western Coll., Wales.	2
March 11,	Bentley Coll.....	3
March 27,	Navigation Coll., Wales....	3
Apr. 17,	Trane, Wales.....	4
May 7,	Littlemill Coll., Scotland...	1
May 11,	Newton Coll., Scotland....	2
May 18,	Newport Coll., Monmouthshire.....	5
July 6,	Barnsley Coll., Yorkshire...	3
July 8,	East Parkhead Coll., Scotland.....	1
July 9,	Cadesby Coll., Conisborough.....	88
Sept 15,	Dudley.....	1

TABLE V.—*Explosions Caused by Gas or Dust, in Great Britain, to Sept.* 15, 1912

Month	Explosions Having Less than Five Fatalities				Explosions Having More than Five Fatalities				Total Explosions			
	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions
January	22	33	1.50	6.61	14	369	26.36	6.39	36	402	11.17	6.52
February	30	43	1.43	9.00	18	920	51.11	8.22	48	963	20.06	8.70
March	27	45	1.66	8.11	19	608	32.00	8.67	46	653	14.19	8.33
April	21	40	1.90	6.31	18	462	25.67	8.22	39	502	12.87	7.06
May	32	52	1.62	9.60	15	667	44.47	6.85	47	719	15.30	8.51
June	24	30	1.25	7.21	13	1,002	77.08	5.93	37	1,032	27.89	6.70
July	24	38	1.58	7.21	15	830	55.33	6.85	39	868	22.26	7.06
August	41	66	1.61	12.31	17	537	31.59	7.76	58	603	10.40	10.52
September	26	37	1.42	7.81	18	747	41.50	8.22	44	784	17.59	7.97
October	34	49	1.44	10.21	23	711	30.91	10.51	57	760	13.33	10.33
November	25	40	1.60	7.51	21	636	30.29	9.59	46	676	14.70	8.33
December	27	42	1.56	8.11	28	1,784	63.71	12.79	55	1,826	33.20	9.97
Totals	333	515	.....	100.00	219	9,273	.....	100.00	552	9,788	.....	100.00
Averages	.....	.....	1.55	.....	.....	.....	42.34	.....	.....	.....	17.73	.....

TABLE VI.—*French Explosions Caused by Gas or Dust*

Date	Place	Number of Fatalities
1814–Oct. 28,	du Garston	1
1817–June 8,	Latour	1
Aug. 7,	Lans Nom	1
1818–Feb. 14	des Souchettes	1
March 13,	Latour	1
March 29,	Latour	1
1823–Aug. 9,	du Chanfour	22
1826–June 29,	St. Jean de Valeriscle	1
Aug. 2,	de la Larade	1
1832–July 2,	de l'Estang	3
Aug. 2,	St. Joseph	10
1833–April 16,	Mamby	1
Oct. 30,	Mamby	17
1834–May 29,	du Port de Lane	1
1835–Aug. 30,	St. Claude	5
1836–July 2,	Osmond	2
July 4,	Osmond	4
1837–Sept. 20,	Osmond,	1
1839–March 6,	du Creusot	1
1842–July 3,	Beaunier	4
Aug. 18,	Charles	15
Sept. 4,	Luce	1
1844–Oct. 15,	du Chene	1
1845–April 16,	Abylon	1
1846–Oct 12,	Villefosse	1
Nov. 12,	de Ste. Barbe	1
1847–Sept. 5,	Fournier	3
1848–April 3,	Fournier	5
Sept. 20,	Bertrand	1
1849–May 7,	Abylon	1
June 30,	de Cherond	1
Oct. 30,	Roche-la-Moliere	1
1850–July 3,	de Villars	5
1851–March 26,	d'Avaire	11
April 25,	Cinq Sous	6
May 9,	St. Jean de Valeriscle	1
July 25,	du Raone	3
Aug. 26,	Marie	1
1852–Oct. 21,	Luce	1
1853–July 9,	Ravez	9
July 29,	Cinq Sous	13
Oct. 30,	Abylon	4
1854–Jan. 8,	Meyron	1
Feb. 6,	du Re Sokil	1
Feb. 6,	Canol	2
July 2,	Abylon	12
Aug. 5,	du Cret de Mars	1
Aug. 13,	La Respie	4
Aug. 25,	d'Agin Court	6
Sept. 2,	Marie Louise	1
1855–Jan. 30,	Charles	2
Aug. 29,	Charles	4
Oct. 22,	Ravez	29
Oct. 26,	Hippolyte	2
Dec. 17,	de Ste. Barbe	1
1856–June 13,	Lans Nom	2
July 5,	du Cret de Mars	1
July 17,	de la Garenne	1
1857–Jan. 26	de la Pompe	1
April 2,	Marie Louise	2
June 4,	St. Mathieu	3
1858–Jan. 20,	Fendue	1
May 29,	Werbrauck	2
Aug. 16,	Cinq Sous	2
Aug. 18,	Delaynaud	1
Oct. 2,	de la Garenne	1
1859–May 14,	du Lud	1
1860–April 6,	Ste. Marie	15
July 30,	Bleuse Bome	1
Aug. 30,	Villars	1
Oct. 15,	Ernestine	1
1861–Jan. 27,	de la Chaux	2
May 26,	du Breuil	21
July 2,	Gallois	1
July 9,	Ste. Amelie	1
July 22,	Grangier	3
1862–March 24,	Beauvais	1
April 10,	St. Urbain	2
May 22,	St. Charles	1
June 1,	Mires	1
Aug. 23,	St. Claude	3
1863–Jan. 26,	Bleuse-Bome	3
May 22,	Fendue de Villars	1
Aug. 24,	Davy	5
Aug. 27,	du Gabet	2
Oct. 28,	Luey	1
1864–Feb. 11,	Monterral	1
1865–Jan. 30,	Beaunier	1
Feb. 1,	de la Chaux	3
Feb. 9,	Ourenne	39
July 17,	Lacroix	1
1866–April 9,	Jabin	4
April 16,	Beauvais	1
1867–Aug. 11,	Fendue de Villars	39
Oct. 12,	Cinq Sous	89
1868–Jan. 3,	St. Augustin	1
Feb. 2,	No. 1 Ostricourt	4
Feb. 19,	St. Martin	3
Aug. 23,	Monterrax No. 1	5
Sept. 25,	No. 1 Lievin	2
1869–March 19,	d'Herin	5
May 21,	Monterrax No. 2	15
Aug. 12,	St. Louis	2
Aug. 21,	Monterrax No. 2	19
Nov. 22,	de la Chaux	3
1870–May 14,	Lucy	1
June 9,	St. Paul	1
July 7,	Abraham	3
July 14,	Lucy	3
July 19,	de la Compe	2
Aug. 21,	Fournier	2
1871–Feb. 1,	Ste. Marie	1
Feb. 21,	Ste. Helene	1
April 17,	de la Garenne	18
June 19,	Jabin	3
July 3,	No. 2 Lievin	2
Sept. 8,	Jabin	70
1872–April 22,	No. 1 Auchy-la-Tour	1
Nov. 8,	Ste. Eugenie	41
1873–Feb. 21,	Le Chanfour	1
Feb. 23,	Charles	1
April 20,	Devillaine	1

French Explosions—Continued

Date	Place	Number of Fatalities
1873–May 15,	Devillaine	1
June 7,	No. 2 Auchy-au-Bois	7
Sept. 17,	No. 3 Lens	1
1874–Jan. 30,	du Raone	1
March 25,	No. 4 Lens	5
July 23,	Davy	2
July 31,	No. 2 Noaix	2
Oct. 12,	Reussite	3
1875–Jan. 3,	du Sosprat	1
Jan. 15,	de Orelys	1
April 5,	Dolonieux	7
May 9,	No. 2 Auchy-au-Bois	2
July 28,	No. 1 Auchy-au-Bois	3
Oct. 22,	Marie	4
1876–Jan. 4,	Jabin	186
Dec. 24,	Malastre	3
1877–Feb. 22,	No. 1, Lievin	3
June 9,	No. 3 Lens	1
1878–Nov. 7,	de l'Ouert	1
1880–Feb. 7,	No. 2 Ferfay-Cauchy	1
April 5,	du Feljas	4
April 5,	Bonnepart	1
April 29,	de l'Echo	2
July 20,	No. 1 Flechinelle	1
1881–April 16,	No. 1 de la Vernade	2
May 25,	No. 1 de Lampret	1
May 17,	du Lavpiat	8
May 28,	No. 1 de Lampret	5
July 29	St. Mathieu	4
Oct. 31,	des Moronniers	1
Aug. 22,	de Leforest	1
1882–April 13,	No. 3 Lievin	8
Jan. 28,	des Rosiers	1
July 12	St. Felix	1
1883–Jan. 24,	No. 1 Courcelles-les-Lens	1
Feb. 12,	No. 5 Lievin	2
Feb. 21,	Ourenne	8
April 9,	Beaunier	1
April 13,	l'Echirreur	4
April 16,	No. 1 Lievin	7
1884–Jan. 5,	Devillaine	1
1884–Jan. 11,	No. 2 Ferfay-Cauchy	17
July 24,	Beaunier	2
Aug. 23,	No. 1 Bully Grenay	1
1885–Jan. 14,	No. 1 Lievin	28
May 21,	de l'Arbonnet	1
June 7,	No. 1 Noaix	3
Aug. 21,	No. 1 Courcelles-les-Lens	10
1886–Feb. 27,	No. 1 l'Escaipelle	1
June 24,	St. Charles	23
1887–March 1,	Chatehes No. 1	79
July 10,	Sud	1
July 10,	Notre Dame	1
Nov. 16,	Montmarte No. 2	1
1888–July 15,	de Molieres	1
Aug. 28,	No. 2 Ferfay-Cauchy	2
Nov. 2,	de Compagnac	49
1889–March 15,	Central	17
May 1,	No. 1 Noaix	1
July 3,	Verpellieux	207
1890–July 8,	Chapelon	1
July 29,	Pelisnier	113
Aug. 4,	Pelisnier	3
1891–March 3,	de Creal	2
Aug. 7,	de Compagnac	1
Dec. 6,	de la Manufacture	62
1895–July 9,	Pontil	1
Sept. 25,	No. 5 Noaix	1
1896–May 6,	No. 1 Vicoigne	1
Aug. 14,	Beneges	1
Aug. 17,	Herin	1
1897–March 5,	No. 1 Droeviert	1
April 6,	Ste. Eugenie	4
July 26,	de l'Arbonnet	1
1900–May 2,	Renard	1
June 29,	de Creal	1
Nov. 12,	No. 3 Bully-Grenay	1
1901–March 21,	No. 2 Droeviert	4
July 19,	de Molieres	9
Aug. 1,	No. 1 Bully-Grenay	1
1902–Oct. 15,	No. 8 de Lampret	8
1903–Aug. 26,	No. 2 Bully-Grenay	2

TABLE VIII.—*Belgian Explosions Caused by Gas or Dust*

Date	Place	Number of Fatalities
1887–March 4,	LaBoule, Quaregnon	113
1891–July 19,	Forchies	27
Oct. 28,	No. 8 de la Lourire	2
1892–Feb. 27,	Belle Vue & Bienvinne	2
Mar. 11,	No. 3 Bois de la Haye	160
May 31,	No. 6 D'Hornee-Wasmes	1
1893–May 23,	Chaivirs	5
Aug. 7,	Chaivirs	1
Oct. 21,	Soxhluse	1
1894–April 1,	No. 7 du Gouppe	3
May 12,	No. 10 de Feni-Raisin	1
May 27,	Viernoy	9
June 2,	Pays-Bas	3
June 29,	No. 2 Charbonnages Rewnis	2
June 12,	Alliance	2
1894–Sept. 9,	Ste. Ive	1
1895–July 29,	St. Alphonse	1
1898–Mar. 25,	Cinq-Gustave	4
May 25,	Crachet No. 12	16
July 3,	No. 7 du Grand Homu	1
Aug. 13,	St. Arthur	8
1900–May 14,	St. Bernard	1
July 12,	Albart	1
1901–April 26,	de Buisson	19
1903–June 5,	St. Jacques	1
1904–Jan. 30,	No. 9 du Gouppe	2
1905–July 17,	Viernoy	16
1908–Jan. 19,	No. 5 Couchant du Fluin	10
Feb. 29,	No. 1 de Ohlin	2
1909–Apr. 17,	Leval	1

TABLE VII.—*Explosions of Gas and Dust in Coal Mines of France, by Months,* 1814 *to* 1904

Month	Explosions Having Less than Five Fatalities				Explosions Having More than Five Fatalities				Total Explosions			
	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions
January	14	18	1.29	8.97	3	231	77.00	6.25	17	249	14.65	8.33
February	15	26	1.73	9.62	2	47	23.50	4.17	17	73	4.29	8.33
March	7	11	1.57	4.49	5	117	23.40	10.42	12	128	10.67	5.88
April	15	31	2.07	9.62	7	66	9.43	14.58	22	97	4.41	10.79
May	15	18	1.20	9.62	4	49	12.25	8.33	19	67	3.53	9.31
June	11	18	1.64	7.05	2	30	15.00	4.17	13	48	3.69	6.37
July	29	56	1.93	18.59	7	368	52.57	14.58	36	424	11.78	17.65
August	22	38	1.73	14.10	10	136	13.60	20.84	32	174	5.44	15.70
September	8	11	1.37	5.13	1	70	70.00	2.08	9	81	9.00	4.41
October	13	22	1.69	8.33	4	143	35.75	8.33	17	165	9.71	8.33
November	5	7	1.40	3.20	2	90	45.00	4.17	7	97	13.86	3.43
December	2	4	2.00	1.28	1	62	62.00	2.08	3	66	22.00	1.47
Totals	156	260	.....	100.00	48	1,409	.....	100.00	204	1,669	.....	100.00
Averages	.....	.....	1.67	.....	.....	.....	29.35	.....	.....	.....	8.18	.....

TABLE IX.—*Explosions Caused by Gas or Dust in Belgium, 1891 to 1909*

Month	Explosions Having Less than Five Fatalities				Explosions Having More than Five Fatalities				Total Explosions			
	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions	Number	Fatalities	Average Number of Fatalities per Explosion	Percentage of Total Explosions
January	1	2	2.00	5.00	1	10	10.00	10.00	2	12	6.00	6.67
February	2	4	2.00	10.00	.....	.....	.....	.....	2	4	2.00	6.67
March	1	4	4.00	5.00	2	273	136.50	20.00	3	277	92.33	10.00
April	2	4	2.00	10.00	1	19	19.00	10.00	3	23	7.67	10.00
May	3	3	1.00	15.00	3	30	10.00	30.00	6	33	5.50	20.00
June	4	8	2.00	20.00	.....	.....	.....	.....	4	8	2.00	13.33
July	3	3	1.00	15.00	2	43	21.50	20.00	5	46	9.20	16.66
August	1	1	1.00	5.00	1	8	8.00	10.00	2	9	4.50	6.67
September	1	1	1.00	5.00	.....	.....	.....	.....	1	1	1.00	3.33
October	2	3	1.50	10.00	.....	.....	.....	.....	2	3	1.50	6.67
November	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
December	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Totals	20	33	.....	100.00	10	383	.....	100.00	30	416	.....	100.00
Averages	.....	.....	1.65	.....	.....	.....	38.30	.....	.....	.....	13.87	.....

TABLE X.—*Production, and Fatal Accidents Caused by Explosions of Gas or Dust in Coal Mines of United States, for Five-Year Periods from* 1870 *to* 1912, *Inclusive*

Period	Production, Short Tons	Total Number of Accidents				Accidents Having More Than Five Fatalities				Average Number of Mines	Average Production per Year per Mine	Accidents per Mine	
		Accidents	Fatalities	Accidents per Million Tons	Fatalities per Million Tons	Accidents	Fatalities	Accidents per Million Tons	Fatalities per Million Tons			Total Number	More than Five Fatalities
[1]870–74	241,582,459	54	86	0.22	0.36	3	22	0.01	0.09	1,998	24,100	0.027	0.0015
[1]875–79	292,171,479	37	83	0.13	0.28	7	45	0.02	0.15	2,862	20,400	0.013	0.0024
[1]880–84	496,776,865	46	369	0.09	0.74	12	315	0.02	0.63	3,118	31,800	0.015	0.0039
1885–89	645,380,403	29	225	0.04	0.35	11	189	0.02	0.29	2,761	46,700	0.011	0.0040
1890–94	858,761,003	81	481	0.09	0.56	18	372	0.02	0.43	3,369	48,000	0.024	0.0053
1895–99	1,059,050,545	119	531	0.11	0.50	24	397	0.02	0.37	4,679	45,300	0.025	0.0051
1900–04	1,573,747,096	163	1,360	0.10	0.86	30	1,152	0.02	0.73	5,985	52,600	0.027	0.0050
1905–09	2,163,900,651	248	2,150	0.11	0.99	63	1,890	0.03	0.87	6,051	71,500	0.041	0.0104
1910–12[a]	1,526,664,275	88	1,126	0.06	0.74	43	1,051	0.03	0.68	6,109	83,300	0.015	0.0070
Totals..	8,858,034,776	865	6,411	.....	.....	211	5,433	.....	.....	.....	.....	.....	.....
Average.	.....	.....	.....	0.10	0.72	.....	.....	0.02	0.61	4,018	51,300	0.023	0.0057

[a]Three years only.

Discussion

George S. Rice, Pittsburgh, Pa.—I think it is very wise to point out the greater liability to explosions in certain parts of the year, if there be such difference, with the caution, however, that vigilance be not relaxed at other times. In comparing the figures for the United States with those of Europe, I think we should separate them as far as possible according to cause of origin and means of propagation. That is to say, it would appear to me that the majority of our explosions are due more directly to the ignition of coal dust directly from the flame of explosives, whereas, judging from my inquiries and observations in some of the mines of Europe, the majority of their explosions have been caused by the ignition of firedamp. Now, in the case of firedamp, I do not think that anybody has yet attempted or has been able to find any connection between the seasons and the issuance of methane into mines. One thing must be borne in mind, that if you get a strong explosion of firedamp, dust of a character that might not ignite readily by itself might assist in the extension of an explosion of firedamp through a considerable area in the mine, if not throughout the mine. While it is true we have many lesser explosions in this country attributable to firedamp, it would appear that the majority of all the mine explosions of the country, in considering the means of propagation beyond the locality of the origin, are due to coal dust; and it is the liability to coal-dust explosions that appears to be affected by seasonal changes. Hence the desirability of separating statistically (if it could be done by the inspectors) firedamp explosions from coal-dust explosions, in order to obtain the true value of the seasonal effect. Nevertheless, in spite of the inclusion of firedamp explosions with coal-dust explosions, the preponderance of the evidence, as indicated in Mr. Eavenson's figures for American mines, has shown that in this country there were more explosions in the winter and spring months, when the mines are drier; that is, when the mines are apt to be drier. Whereas, according to the European figures that he has gathered, it seems to be very clearly demonstrated that in France, England, and Belgium, the time when explosions occur has no connection with any particular season. That, I think, is partly due to the point just mentioned, that a majority of their explosions have originated from firedamp. The European mining men have been more careful, generally speaking, with the use of explosives than we have, so that the cause of ignition from that source has been less. Only comparatively few of the mines in foreign countries permit the use of black powder. It is not permitted at all in France, Belgium, and Germany, except in a few of the shallow, non-dusty and non-gaseous mines. In Belgium, in mining coal in gaseous or dusty mines no explosive is used, and in rock work only permissible explosives may be used; but it must be admitted the non-use of explosives

in coal is not a handicap in coal getting, since the mining is by the long-wall system and the roof weight brings it down easily.

Apart from the generally accepted fact that in explosions in European mines, firedamp, either wholly or in part, plays a most important rôle, an additional argument may be made as to why the seasonal changes appear to have so little effect in European coal mines, as seems to be indicated by Mr. Eavenson's figures, viz.: The European mines are so much deeper than American mines, and hence so much warmer, practically always warmer than the air at the surface, that there is little difference in the drying effect of the air current in either summer or winter. That is, the temperature of the air current by the time it has reached the working places has been so raised by the natural heat of the strata that it is always drying in effect even in summer time, and precautions are taken accordingly. On the other hand, in America the temperature of the mines is so much cooler, that in summer it is less than the average outside temperature, hence the entering air current, through being cooled, becomes fully saturated, and this in turn moistens the walls and ultimately the dust. Therefore there is for a few months lesser liability to have true coal-dust explosions.

C. M. Young, Lawrence, Kan.—A few years ago the State Geological Survey of Kansas collected what was believed to be all the available information concerning coal-mine explosions, including causes, effects, and dates. This information was gathered from all parts of the world where records have been kept. After the statements of figures and facts had been tabulated, copies were sent for correction to the various countries from which the data had been obtained. When these corrected copies had been returned, it was considered that the list of explosions was as nearly correct as it could be made.

From the data in these lists it would be concluded, as Mr. Eavenson states, that there is no necessary connection between seasons and explosions.

Some possible reasons may be suggested for the increase in explosions mentioned in the paper. First, the change in the character of mine labor, with the partial disappearance of the miner who looked upon his labor with pride and had the attitude of the skilled artisan toward his work. Second, the increased use of explosives. It is true that the use of permissible explosives is increasing, and this fact decreases the danger, but there are still many districts in which black powder is used and its use is likely to continue for a long time. In many districts the ratio of the amount of powder used to coal mined is increasing. The tendency to depend almost entirely upon explosives for breaking down the coal leads to the use of excessive amounts of powder. Third, the increasing tendency to shoot off the solid.

It is possible that one reason for the occurrence of many explosions

outside the winter season may be found in the belief that they will occur almost exclusively in the winter. The belief that there is most danger of the occurrence of explosions in the cooler seasons may lead to carelessness in other seasons. In February and March the miner expects explosions, if there is any possibility of their happening, and at most other seasons he thinks that he is fairly safe, and it is possible that he is then a little less careful than he is in the winter months because of this belief.

H. H. Stoek, Urbana, Ill.—There is possibly another explanation or reason why we apparently have more explosions in winter than in summer, because our mines then work much more actively in many parts of the country. The European mines work more uniformly than ours, so that we might expect on that basis to have the explosions distributed throughout the year. Then there are some who think that when a mine is working irregularly it is more liable to explosions than when it is working regularly. Has Mr. Eavenson any figures showing connection between explosions and earthquakes?

Mr. Eavenson.—No, I have had no experience in the earthquake line, and I am mighty thankful to say I have had very little experience in the explosion line.

William Griffith, Scranton, Pa.—I think a question worthy of scientific investigation is whether the diffusion of gas in mines is in any way affected by the astronomical condition like tides in the ocean. In my experience, passing through the anthracite mines, I have for some time made it a practice to ask the inside men, fire bosses, mine foremen, and others, what time of day the most falls of roof occur, and I find that the great preponderance of roof falls in the mines occur at night time, *i.e.*, when the sun is on the other side of the earth. The question has arisen with me whether there is any possible effect from the attraction of the sun or moon, which would cause such a thing to occur or to effect the efflux of gas.

R. D. Hall, New York, N. Y.—People fail to realize, in comparing one country with another, that there must always be differences between them. England and Scotland lie against a great ocean, receiving all the time moist air with a percentage of saturation which often amounts to 100. That degree of moisture content is quite rare in the United States, and, moreover, the temperatures in Great Britain never get so low as those on this side of the water. Consequently, the air on entering the mines does not lose saturation so markedly there as it does here. This makes the danger of coal dust less marked, and as it is the coal-dust explosion which is seasonal, there is not likely to be the seasonal difference in accident frequency in Great Britain which we have here.

Another cause for decreased danger from coal-dust explosions in Great Britain is the use of the long-wall system. When an explosion of any

kind takes place in such workings there is plenty of room for the explosive violence to expend itself. It is rarely found in Great Britain that the violence is marked near the coal face. The explosion seems usually to start and work its principal havoc in the tightly gob-packed headings. Consequently the frequent use of the long wall probably decreases the frequency of coal-dust explosions.

And conversely long wall increases the probability of gas explosions, for the ventilation is frequently rendered inadequate where that system of mining is in operation.

But in regard to seasonal variations, we must not forget that the winter in Europe sets in earlier than here. In fact, it is quite usual in Great Britain to regard the last three months of the year as winter. Here the first three months are generally so classified.

W. H. Grady, Bluefield, W. Va.—One of the speakers made the point that perhaps the increase in explosions was due to the difference in supervision, due to the fact that they are employing Hungarians and Poles, rather than English and Welsh miners. If that is true, and I believe it is, the proper thing is to supply them as much supervision as that difference requires, and some mine operators have recognized the need of more and better supervision and are supplying it by safety foremen, patrolmen, or assistant foremen, or whatever they call them. The particular instance that has come under my observation is that of the United States Coal & Coke Co., at Gary, W. Va., where the death rate per 100,000 tons is one-fifth of what it is in the Pocohontas field. In speaking with different operators on that point, they sometimes say that increased supervision brings about increased cost; but with increased supervision, it is found that increased efficiency follows, and rather than an increase of cost, we have a reduction in cost and a reduction in the death rate.

R. V. Norris, Wilkes-Barre, Pa.—In connection with increased supervision, recently in the anthracite fields various large companies have been putting in a safety patrol system.

E. B. Wilson, Scranton, Pa.—For the benefit of Mr. Stoek, we had an earthquake in Scranton some months ago, and I took occasion to write to all the operators to find if there had been an increase in gas, and they said no; that on the contrary, there had been less. I attribute that to a high state of the barometer.

F. Z. Schellenberg, Pittsburgh, Pa.—I believe explosions are apt to occur on Monday morning or after a holiday, when there is irregularity of supervision, or of the immediate examination of the working places. Some of the men who have had that responsibility are out, which is the cause to my mind of one of the largest catastrophies that we have had here.

S. A. Taylor, Pittsburgh, Pa.—A question has been in my mind for several months. It was stated to me by a man who had just returned from Europe, that a large explosion that took place in France last spring was caused by electric currents. On investigation, it was found that it was due to the electric waves caused by a wireless telegraph. Talking this matter over with one of the professors of mining in one of the schools not far distant from here, he told me that they had had something of the same experience, in connection with their wireless telegraph plants, and they were obliged to stop the operators. They ascertained that the two wireless stations set fire to a building which was half way between the two. Now, if anything can be evolved along that line, it seems to me we are treading on very dangerous ground when we allow the promiscuous installation of wireless telegraph systems. It seems to me it should be investigated by those who have the ability and power to do so.

Howard N. Eavenson.—I think Mr. Rice's point, that if explosions could be separated into those caused by gas and those caused by dust, more dust explosions would be shown to have occurred in the winter months than in the summer months, is well taken. He knows, however, better than I do, the trouble one has in trying to separate these two classes of explosions from any records we now have. To illustrate this: I know personally of an explosion that was ascribed to dust, while it was a well-known fact in the vicinity that gas could be exploded by any miner in his working place any time after firing a shot.

There was no intention in writing this paper of making a comparison of the explosions due to seasonal variations in the different countries. My only idea was to present the facts.

As to Mr. Griffith's point about falls of roof: We prepared statistics for our 12 mines, covering the past 10 years, classifying accidents by the different days of the week, different hours of the day, and the causes—whether falls of coal or slate. The average number of accidents per day was practically the same throughout the week. There were a few more between 10 and 11 o'clock than in any other hour in the day. This does not exactly cover his point, as there might be many more falls at night than in the day time, when no accidents would result from them.

Regarding the explosion caused by wireless: The writer knows of a mine that had been shut down for about three years, which generated gas in small quantities. The fan and pumps were run two or three days once a month and were shut down the rest of the time. One afternoon during an electric storm there was an explosion in the mine, although there was nobody working in it at the time nor had there been for 18 days. The pumps were not running and there was no fire under the boilers. There was no electric current in the mine, as the trolley-wire connections were broken at the foot of the shaft. A careful investigation

was made and the only cause to which this explosion could be ascribed was lightning, which probably ran down the shaft, possibly through the ropes, thence to the cage and to the tracks and followed the rails in to the place of the explosion. This place was an excavation being made for a stable when the mine was shut down, and which was driving up hill and was a dead end. Gas had evidently accumulated in here, and as the switches were all set to go directly to this place it was evident that the current had followed the rail into the gas and had then exploded it.

Richard Peters, Jr., Uniontown, Pa.—I was connected with a mining operation in Alabama a few years ago. It was shut down on July 4, 1910, for the usual holiday, and occasion was taken to make some repairs on the fan. Along about 3 o'clock in the afternoon a sharp flash of lightning struck the steel head frame on the shaft. A few moments later black smoke came out of the shaft mouth. The fan was started up and turned over slowly to see what effect would be produced, and it was noted that the smoke immediately cleared away. After turning air down into the mine a party was sent down, and upon investigation found that the lightning flash had gone down the steel work of the shaft and had followed the trolley wire to a point in the mine where it short-circuited and ignited a pocket of gas. This was evident, as the damage done by the explosion was local.

Mr. Norris.—There have been two similar cases in the anthracite region of gas fired purely by the electric spark from a storm.

Mr. Rice.—Regarding the difficulty that Mr. Eavenson mentions of getting accurate statistics, I think it is particularly true in the case of explosions. At the present time mine-accident statistics are collected by the various State inspection departments. The U. S. Bureau of Mines accepts these figures in its compilations for the whole country. The State inspection departments have no uniform system in that matter, and while some inspectors will call an explosion, the propagation of which is due to dust, a dust explosion, others will classify it on the basis of its origin. It is quite a usual thing in the Middle West to call an explosion a "windy shot," even if it has extended to the shaft. That is the cause of one of the difficulties we have in analyzing results. In statistics which are furnished to the Bureau of Mines, I have found many instances where what I would term a dust explosion has been called a gas explosion, windy shot, or powder explosion. I think all should try to obtain uniform methods in reporting; and, as regards explosions, I think we should classify them on the basis of their chief means of propagating, that is, gas or dust, giving the origins in a separate column under these classifications: Blown-out shot; overcharged shot; ignition of fire damp; electric arc or grounding; open flame; gob or mine fire.

The Appraisal of Coal Land for Taxation

BY H. M. CHANCE, PHILADELPHIA, PA.

(Pittsburgh Meeting, October, 1914)

WITHIN the last 10 years the subject of mine taxation in its relation to coal-mining interests has come to have growing importance, not only to those engaged in the mining of coal, but also to the owners of land underlain by coal. A disposition has developed among legislators to accept the principle, which has been laid down by many judicial decisions, that coal underlying the surface must be regarded as a part of the real value of all properties so underlain, and as such is chargeable with such burdens of taxation as the necessities of the government may impose. These necessities arise from the normal expense of maintaining State and local governments, and in the past have been met principally by taxes based upon assessments of real-estate values, which values in many cases have been confined principally or wholly to the value of the surface, without taking into consideration the value of minerals underlying the surface.

It is not proposed here to deal with the question of Federal taxation, whether levied on production, or on products transported from one State to another, or exported from the country, or upon the gross or net earnings of corporations engaged in interstate commerce, or upon the production of materials which may be transported from one State to another, or levied upon the profits or upon the obligations of corporations. The present discussion will be confined to the appraisal of coal lands as assessed for State, county, township, or borough taxation.

The methods heretofore and at present in use in the different States vary widely and have resulted in more or less litigation. Discussions appearing from time to time in the technical and trade journals and in the proceedings of mining and engineering associations frequently direct attention to the need for the adoption of fixed principle in levying taxes, and of uniform methods for applying such principles, but there does not seem to be a general demand from the public for the adoption of such methods.

The greatest burden of taxation must always be from taxes levied to meet local requirements, because the maintenance of schools, of the poor, of roads and bridges, of local courts, of prisons and hospitals, of the police, of roads and highways and their lighting, and of sanitary super-

vision, will always cost far more per capita than the cost of maintaining the Federal or State governments.

As taxes for local purposes are collected under authority conferred by State legislation, and as the constitutions of the several States contain provisions concerning the rights of persons and property which require uniformity in methods of taxation, and as the courts have generally affirmed the principle that coal, if of material or determinable value, must be regarded as assessable and taxable property, differing in no respect from other forms of property, this principle is supposed to underlie the methods in general use.

In connection with matters relating to the application of this principle in the determination of taxable values, an investigation was made recently to learn as fully as possible the details of present practice in the States in which coal mining is an important industry. This inquiry resulted in the compilation of a large mass of data, giving perhaps a better insight into these conditions than anything that has yet been available. It disclosed the fact that few of the States have definitely adopted the principles upon which mineral (or coal) land shall be appraised for the purposes of taxation, and that few States have uniform methods applying to all parts of the State for fixing such values.

The investigation upon which this article is based included the principal coal-mining portions of Alabama, Arkansas, Colorado, Illinois, Iowa, Kansas, Kentucky, Missouri, Montana, Ohio, Pennsylvania, Tennessee, Texas, Utah, Virginia, West Virginia, and Wyoming. It involved correspondence with about 350 State, county, and township officials engaged directly or indirectly in the assessment of the value of coal land for taxation, and includes more or less useful and authoritative information obtained from about 120 of these officials.

While the inquiry disclosed the fact that uniformity in making such assessments is in practice almost unknown, entirely different methods often being used in adjoining townships in the same county, or in adjoining counties in the same State, it did, however, clearly indicate that in nearly all of the coal-mining States certain broad principles were recognized, and these tend to establish a basis to which future practice will doubtless be made to conform. While these principles, however, are theoretically assumed as governing factors in fixing the basis for assessments, the methods by which they are applied, or are attempted or supposed to be applied, present extraordinary variations and inconsistencies. This condition, in some cases, has led to attempts to remedy such inconsistencies by the creation of State commissions, or boards, or courts, charged with the duty of equalization.

The principles that appear to have received most general recognition, and the exceptions to the same, are as follows:

1. That the coal, or coal-mining right, when not owned by the owner of the surface, shall be assessed as the property of the individual or corporation claiming ownership. Exceptions to this rule are found in Kansas, Kentucky, Iowa, Illinois, Missouri, Michigan, Utah, Wyoming, and possibly some other States, such exceptions generally applying to localities in which the coal, through lack of knowledge as to its thickness, quality, etc., has no definable or appraisable value. This principle has been affirmed and reaffirmed by judicial decisions in many States.

2. When the coal and surface are owned by the same individual or corporation, the assessed value is usually fixed at a sum per acre supposed, intended, or assumed to cover the value of both coal and surface; assessors rarely attempting to assess the value of the coal and surface separately. The exceptions to this practice are found principally in those States and districts in which coal has the largest values; examples of such exceptions are found in Pennsylvania, Kentucky, Ohio, Virginia, and Wyoming.

3. Coal, or the coal-mining right, is generally assumed to be assessable at a value representing the price obtainable at voluntary sale, or at a definite percentage (ranging all the way from 20 up to 80 per cent.) of such value. While the legal validity of this principle has been sustained by many judicial rulings and it is presumed, or assumed, to be applied in many States, exceptions are found almost everywhere. Some of the Western States have substituted for this principle, methods of fixing the taxable value based partly or wholly upon the value of the output, or upon the net or gross returns, or upon an arbitrarily assumed value per ton of such output. Thus in Wyoming the value of the output (fixed at from 60c. to 95c. per ton) determines the assessable value. Exceptions to the application of this principle are found in Alabama, Arkansas, Illinois, Iowa, Kansas, Kentucky, Ohio, Pennsylvania, Tennessee, Virginia, and West Virginia, the exceptions being most numerous in localities where the owners of the surface (farmers) still own the coal. In such localities the owners of the coal, generally being anxious to sell it, and believing that high assessments of value will tend to retard sales, usually succeed in keeping the assessed values far below the price obtainable at voluntary sale.

4. Great differences exist both as to principle and practice in assessing the value of coal leaseholds; that is, the equity of a lessee in the coal which he controls under a lease. The equity of a lessee in the coal is not assessed as taxable property in Alabama, Arkansas, Colorado, Kansas, Missouri, Utah, and Virginia. Such equity may be assessed as taxable property in Illinois, Iowa, Kentucky, Tennessee, Ohio, Pennsylvania, West Virginia, and Wyoming, but the practice is not uniform throughout any one of these States; in the aggregate only a comparatively small number of leaseholds being assessed for taxation. The whole value of

coal held under lease is usually assessed to the owner of the property, the equity of the lessee being disregarded. This practice, in many cases, doubtless originates in the difficulty experienced by assessors in attempting to determine the value of the equity of the lessee, and also because the assessors assume that as the coal value (in such case perhaps being determined by the royalty per ton paid by the lessee) is assessed to the owner of the leased premises, and the value of the improvements owned by lessee is assessed to the lessee, the operation as a whole is thus bearing its full share of the burdens of taxation. Cases in which the terms of the lease provide that any increase in the taxes assessed after the execution of the lease shall be borne by the lessee, also tend to prevent the assessment of leasehold equities separately from the coal value assessed to the owner of the land. On the other hand, leases which are so drawn as to constitute a sale of the coal in the ground, tend to encourage assessors to assess a portion of the coal value to the lessee as the owner of the same.

5. The proposal to base the assessment of coal value arbitrarily upon the quantity of coal (as defined in foot-acre or other units) and a fixed value per ton or per foot per acre, to apply uniformly over a prescribed district, is steadily losing ground, probably because it does not seem to be in conformity with the principle laid down by the courts, that assessments must be fixed by the price obtainable at voluntary sale. This proposed method, therefore, is hereafter likely to be used only as a auxiliary means of determining the voluntary sale value, and not as a method for directly fixing the assessable value.

Many officials are inclined to be pessimistic as to the possibility of securing greater uniformity in methods of making assessments, apparently because the subject appeals to them, and to the residents of each district, more in the light of a local issue than of one of broader importance. Some of the suggestions illustrating the trend of thought as to proposed changes and the securing of better or more satisfactory results may be of interest thus:

"By making the tax directly upon the output."
"Now being considered by the State Tax Commissioners."
"By revising method of taxation for all property."
"By estimates of quantity of coal in each tract."
"Most any other way would be an improvement."
"Revision of laws to provide assessment at fair sales value."
"Revison of laws to determine assessments by amount invested."
"By assessing surface and coal separately."
"Revision desired, but no prospect of getting revision."
"Tax should be on the production."
"State officials now considering method, but no conclusions reached."
"Should be controlled by a Tax Commissioner."
"No one seems to be thinking about it, are satisfied with present methods."
"By the appointment of County Tax Commissioners."

"Assessments should be low enough to permit operators to hold (unworked) coal land."

"By classifying coal land as working, reserved and undeveloped acreage."

"Depth, length of time to reach, accessibility and other conditions should be considered."

"By assessing each field separately at unit prices governed by the average selling price."

"By employing engineers to assist Tax Commissioners."

"By distinguishing between coal to be mined soon and that not to be operated for some time."

"By assessing all coal lands at their commercial value."

"By taxing on an output basis."

"A tonnage tax is inequitable and should not be favored."

"Opposed to a tonnage tax."

"According to output, value of plant and profits, all to be considered in fixing assessment."

"Should be fixed by the net profits per ton."

"Mining engineers should be employed to assist in making assessments."

"There should be a better way found; present methods most unsatisfactory."

"By classification of lands and the actual production."

Of those officials who were willing to express a definite opinion, about 85 per cent. believe that while the methods are faulty, the results are generally satisfactory to land owners, operators, and the general public; about 10 per cent. believe that both methods and results are unsatisfactory; and about 5 per cent. think that the methods and results are partly satisfactory. It will be understood that these are the expressed opinions of officials engaged in assessing coal land for taxation and may not represent the opinions of coal-land owners and operators. An explanation of the statement that the assessments as made are generally satisfactory to owners and operators is found in the fact that in many mining districts the assessors are influenced in greater or lesser degree by the views and desires of a majority of the residents of the district, and adjust the basis for making assessments to meet the approval of a majority of those most interested. This condition accounts for the wide divergence in methods and results often noted in adjacent districts, and again suggests the conclusion that the assessment of coal land for taxation is at present largely regarded as a local issue and is determined independently in different districts by considerations of local expediency.

That the values at which coal land or coal-mining rights are actually assessed for taxation in the principal coal-producing States seldom bear any fixed relation to the value as determined by prices realized at voluntary sale, is illustrated by the following condensed summary:

County Assessments

State		Highest Assessed Value of Coal, Not Including Surface, per Acre	Lowest Assessed Value of Coal, Not Including Surface, per Acre
Alabama	"Supposed to be assessed at 60 per cent. but rarely is assessed at more than 25 per cent. of value."	$1.00 to $ 40.00	$0.12 to $3.00
Arkansas	"Supposed to be assessed at 50 per cent., or less, of value at voluntary sale."	5.00	2.50
Colorado	"Depends on accessibility to railroads."	60.00	25.00
Illinois	"Usually at about 20 per cent. of voluntary sale value."	75.00	2.00
Iowa	"Undeveloped lands not assessed as coal land."	10.00 to 30.00	6.00 to 25.00
Kansas	"Supposed to be at its market value."	20.00 to 60.00	5.00 to 10.00
Kentucky	"Assessors often adopt statement of owners as to value." (Data not complete.)	2.00 to 15.00	1.00 to 4.00
Ohio	"Attempt to approximate value." "More guesswork than anything else." "Actual values tried to be ascertained."	20.00 to 80.00	10.00 to 20.00
Pennsylvania	Methods vary greatly. (Bituminous region only.)	10.00 to 900.00	5.00 to 50.00
Tennessee	Data incomplete	20.00	3.00
Utah	Data incomplete	20.00	10.00
Virginia	At fair market value as per Act of March 7, 1912.	100.00 to 500.00	1.00 to 8.00
West Virginia	"Supposed to be at voluntary sale value."	6.00 to 180.00	3.00 to 15.00
Wyoming	"On net value of output." (Data incomplete.)	20.00 to 30.00	20.00

This summary is not thought to be complete, or free from error. Many of the wide variations in assessments, perhaps most of them, merely reflect actual differences in the real values of coal land. Individual cases in which the assessments are higher or lower than those given in this summary doubtless occur in each of the States above enumerated. The table is introduced to show the maximum assessments in the two counties that show the highest and lowest maximum value and the minimum assessments in the two counties that show the highest and lowest minimum value, for the counties of which a record is available.

Discussion

R. V. Norris, Wilkes-Barre, Pa.—In his *résumé* of coal-taxation methods Mr. Chance has apparently confined his investigation to bituminous coal, so a summary of the anthracite situation as to coal assessment should be of interest.

The anthracite region of Pennsylvania in its 484 square miles of measures lies in the counties of Wayne, Lackawanna, Luzerne, Carbon, Schuylkill, Northumberland, and Dauphin, and in each of these different methods of assessment have been adopted.

The assessments for taxation in Pennsylvania are made under Acts of the Legislatures of 1841 and 1842, under which the assessors are required to "assess rate and value every subject of taxation according to the actual value thereof, and at such rates and prices as the same would bring at a *bona fide* sale after due notice." Under this general law coal land has been assessed as such, but up to about 25 years ago the coal was assessed at rates so low as to be unimportant, though the rates in each township and municipality, even in the same county, showed no agreement or ascertainable basis.

In the late '80s, the need for greater revenue drew attention to the possibilities of collecting much larger taxes from the coal lands and, spurred by public demand, the assessing authorities in some counties undertook a reassessment of coal. The method pursued was to demand from each owner a statement of the area and average remaining thickness of coal on his lands. On this return, made in some cases under oath, an assessment based on foot-acreage was levied, with a resulting great increase in valuation, but the valuation per foot-acre was low and no serious objection was made. This assessment with occasional increases of rate remained in force until 1907, when, in the triennial assessment of that year, a complete revision of the valuation was again attempted, engineers were employed by the counties to check the accuracy of the returns from the owners, and, based on the foot-acreage thus obtained, an enormous increase, to $65 per foot-acre, was made in the valuation. This assessment was resisted in the courts by appeals resulting in very extensive litigation, and a final determination of the validity of the 1907 and all subsequent assessments has not yet been obtained. In the first appeals tried the rate calculated on a foot-acre valuation was reduced by the Luzerne County court to about $45, while the Lackawanna County courts reduced a larger rate to $67. The county authorities were, however, dissatisfied with these results and have since tried other appeals from the same 1907 assessment, but these latter suits have not yet been decided even by the county courts.

Since the guess valuation of a generation ago was given up, four methods of assessment have been attempted or suggested.

First: Valuations based on actual sales.

Second: Valuations based on foot-acres of coal remaining in the ground or remaining available.

Third: Valuations based on royalty values.

Fourth: Valuations based on capitalized estimated profits.

Of these methods, the first, valuations based on actual sales, is indicated as the only strictly legal method by various Pennsylvania Supreme Court decisions. Of the foot-acre method, in Reading Appeal (Pennsylvania Supreme Court Reports No. 229, p. 465), the court says "that the foot-acre value for ascertaining valuation of coal lands of the appellant for the purpose of taxation is not a proper measure of their value;" while the court disposes of valuations based on royalty, and by implication those based on profits (Pennsylvania Supreme Court Reports No. 299, p. 470):

> "Its market value is its fair selling value for cash, not payable as royalty strung out through a long series of years, but payable at the time or as soon thereafter as the value could be determined. Such a method does not make allowance in undeveloped territory for the length of time coal may lie in the ground unmined, undeveloped and unprofitable. It is impossible to reduce to a scientific basis and to mathematical precision the elements of value entering into the present selling price of a tract of coal land. The question is not what earning power coal lands may develop in the future, but what they are actually worth in the market at present."

The sales method has been used both in justifying and in attacking the 1907 anthracite valuations and has resulted in the introduction into evidence of a large number of "sales" of coal land, generally the coal under very small tracts, with but few sales of acreages large enough for operation, and none of this size except sales of operating collieries where the value of the lands could not readily be separated from the value of the improvements and colliery plant. The sales shown have indicated values from $200 or $300 up to $10,000 per acre, with an average for the best lands containing beds aggregating 60 ft. of coal of about $3,000 per acre or $50 per foot-acre, equivalent to about 5c. per ton for the coal available for shipment, estimated at 1,000 tons per foot-acre.

In the face of this, the 1913 assessment has been raised throughout the anthracite region. The rates determined by a foot-acre calculation, but returned in gross price per acre to evade the Supreme Court ruling, are as follows:

	Per Foot-Acre
Wayne County	$67
Lackawanna County	175
City of Scranton	300
Luzerne County	250
Northumberland County	48 to 61
Schuylkill County	9.84 to 68.52
Dauphin County	10.00 to 16.58

While practically all of these have been appealed from, the Lackawanna and Luzerne County and Scranton City rates are absolutely confiscatory; in one case, where under the old valuation the taxes paid by the land owner equaled one-fourth of the average royalties received, the valuation was raised 563 per cent., making the lands worse than valueless to the owners.

Then, considering the various methods of coal-land taxation suggested, we find serious objections to all.

First: *Sale Valuation.*—Impracticable, from the impossibility of finding sufficient sales of typical lands for a basis of value, and further

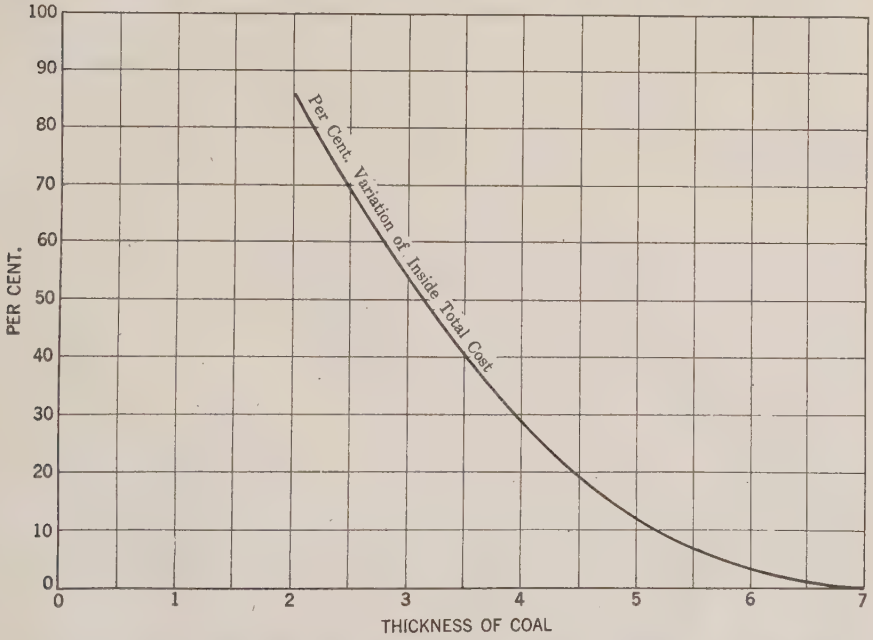


FIG. 1.—RELATIVE COST OF MINING COAL BEDS OF VARIOUS THICKNESSES.

unfair in not considering the actual amount of coal in separate pieces of land, the sale value of a large tract of thick coal being entirely different from the selling price of a tract containing only thin beds, or beds partly exhausted, and the impossibility of separating the value of the coal from the value of improvements and developments in the case of the sale price of operating collieries.

Second: *Foot-Acre Valuation.*—An illegal method, fairer than sale valuations, but presenting difficulties in determining a proper basis of valuation, and offering to unscrupulous politicians too easy a method for flat advances in valuation. This method fails to discriminate in the character of the coal assessed, not only in the thickness of the beds but in their constitution, whether large benches of clean coal or coal laminated

with slate and bone in thin layers and of relatively little value. How much the thickness of beds may affect their value is shown by Fig. 1, the relative cost of mining various thicknesses of beds, compiled from averages of the actual costs in a large number of collieries in the Wyoming field.

Third: *Royalty Valuation.*—Illegal, and unfair from the fact that the rate of output and time of mining are overwhelming factors in the valuation. For example, assume five exactly similar properties, each containing 2,000,000 tons of coal, to be worked out seratim at an average of 100,000 tons per year, and each paying 30c. per ton royalty, an annual royalty paid during the mining of each tract of $30,000; on the basis of Luzerne County, Pennsylvania, 1913 assessment, these would each be valued at $400,000, and would pay approximately $8,000 annual taxes up to the average time of exhaustion. Their present values, on a royalty basis, calculated at 6 per cent. would be as follows:

Tract	Start Mining, Years	Complete Mining, Years	Present Value Royalties	Less Present Value Taxes	Net Present Value
First	0	20	$344,100	$58,880	$285,220
Second	20	40	107,360	110,120	−2,760
Third	40	60	33,550	126,100	−92,550
Fourth	60	80	10,430	131,550	−120,650
Fifth	80	100	3,250	132,550	−129,300

Thus, even without any taxes, the coal held for future necessities is shown to have but small present value, and including the taxes to be paid, except the first assumed tract, to be mined out within 20 years, is worse than valueless, and would be held by the owners at a loss.

This condition may be well illustrated by Fig. 2, showing the present value, and present value of tax charge on $100, to be paid at a future date, as on lands held for reserve.

Further, valuations on such a basis would result in absolute confiscation of lands leased on the low prevailing rates of the past, when the owners accepted royalties of from 8c. to 25c. per ton for prepared coal only, at present about 65 per cent. of the entire production, and further covenanted to pay all taxes assessed against the lands.

Fourth: *Valuation on Estimated Profits.*—This method, which has been proposed but not used in the anthracite region, is manifestly illegal and most unfair because an accurate estimate of future profits is manifestly impossible; profits are dependent not only on the property, but on its management and on the rate of output, the latter at least partly determined by market conditions. That the rate of output is an important factor is shown by Fig. 3, showing the percentage variation in cost in a group of anthracite collieries under varying conditions of rate of output. Further, such a method of assessment is opposed to all ideas

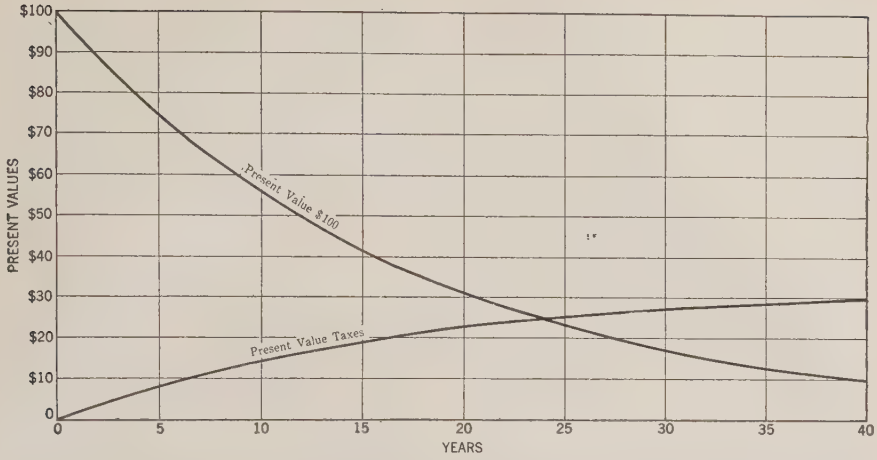


FIG. 2.—PRESENT VALUE OF $100 FUTURE PAYMENT AT DIFFERENT PERIODS AND PRESENT VALUE OF TAXES ON SAME AT 20 MILLS FOR SAME PERIODS, AT 6 PER CENT.

of conservation, as it penalizes good management and good mining by taxing brains as much as coal.

Conclusions.—It appears that none of the suggested or attempted methods of assessment has resulted, or can result, in an equitable valua-

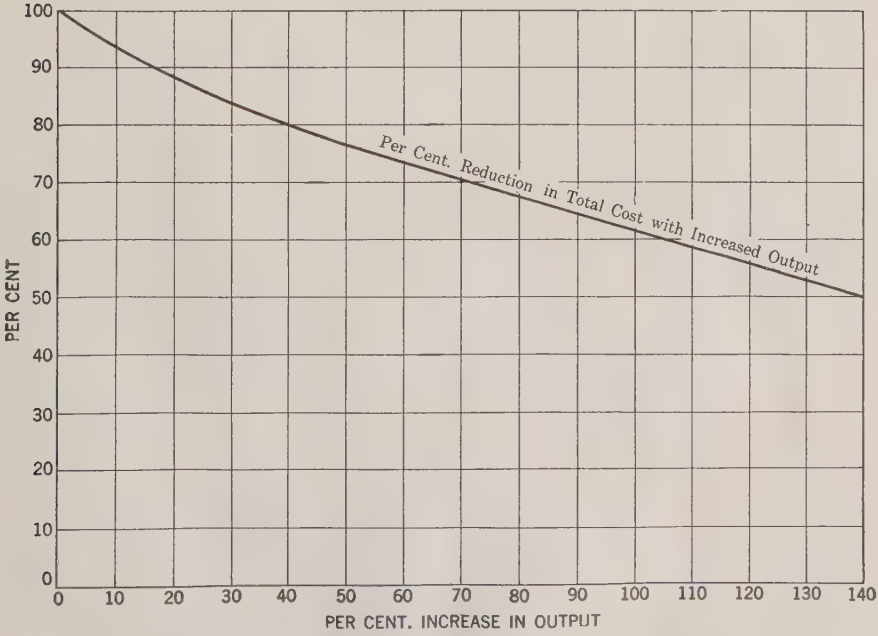


FIG. 3.—VARIATION IN COST OF MINING AT VARYING RATES OF OUTPUT.

tion, fair and just to both the public and the owners of coal land; that even moderate taxation of the coal in the ground is opposed to all principles of conservation, as its effect is to put a tremendous premium on rapid mining, almost regardless of ultimate recovery, to encourage the destruction of poorer and thinner beds interstratified with the better ones, on account of the enormous penalty entailed in slower mining, and to discourage, by prohibitive penalties, the holding of lands in reserve for the future necessities of the people.

For these reasons it appears that the taxation of mineral in the ground is logically and economically wrong, leading to the rapid and uneconomical exhaustion of the mineral wealth of the country, and putting a premium on premature and wasteful exploitation, and that the proper method of taxation for all minerals would be a tax based on the value at the mine of each year's product at the local rate of taxation assessed for that particular year, including an assessment on surface lands, outside improvements, and machinery, the values of which are readily ascertainable, but not including any valuation of mine openings or inside improvements which are incidental to the mining process and which after the exhaustion of the mineral are of no value.

Thus, a colliery producing 1,000,000 tons of anthracite in 1912, with a value at the mine of say $2,500,000, and with surface and improvements valued at $1,500,000, should pay taxes for the year 1913 on $4,000,000 valuation, regardless of the area of coal land tributary to such a colliery, and if for any cause the production in some later year should fall to $1,000,000 in value, and the value of the surface and improvements decrease to $1,000,000, the taxes for the next year should be assessed on but $2,000,000 valuation.

This suggested method of assessment and taxation of coal lands is of course clearly illegal under the present laws of the State of Pennsylvania, and would require special legislation to put it in force, and while not absolutely just, in that it assesses coal from thin and impure beds, costly to mine, at the same rate as that from the more cheaply mined thick and pure beds, it would, if legalized, possess the inestimable advantage of doing away with all uncertainty and litigation as to assessed valuations; result in the payment of taxes in greatest amount at the times of greatest production and consequent greatest population and public need for money; and by concentrating taxation on the land most actively worked, and relieving reserve land from its present crushing burdens, would tend to the conservation instead of the dissipation of the irreplaceable coal resources of the country, by encouraging the complete mining of lands once opened, including all workable beds, large and small, rather than the opening of the best beds on all lands to obtain immediate returns and avoid the burdens of accumulating taxation, even at the cost of the destruction of smaller and less valuable beds lying above the larger ones.

WILLIAM GRIFFITH, Scranton, Pa.—I have not read Dr. Chance's paper yet, but I desire to express myself as being entirely in accord with Mr. Norris's conclusions; also to add that not only is his method of taxation illegal, but there is no other sensible method of taxation of coal or any other mineral land in Pennsylvania that is legal. The only possible method you have now is to send the "ice man" on the property and ask him to guess a valuation on it. There is no possible method that any engineer, or anybody else who has any idea of the value of minerals, would use that would be sustained by the Supreme Court, because the law as it now stands does not recognize mineral value or any method of determining it beyond a mere guess. But the court and owners insist on appointing engineers to do this work and thereby warrant the inference that they desire it to be done by sensible men in an engineering manner. But after it is done, if the engineer in his report explains how he did it, it will be thrown out as being illegal under this antiquated law. That is the reason why we did not state in our Northumberland County report how we did it. We examined 100 or 200 different anthracite properties (there were two engineers and a layman on the Commission). In estimating the value, we used all the methods that Mr. Norris has referred to. We estimated the size, the quantity of coal in the ground, extent and condition of mine workings, its accessibility, its remoteness from market, the royalty, etc., and by combining all these things, and capitalizing the royalty for the length of time we expected the mine to last at a reasonable output, making proper adjustments in every case to meet varying conditions, we finally arrived at what we believed would be a reasonable value for each separate piece of land. But in our report we simply stated the value of each tract, giving the number of acres, without saying anything about how we got at it. And that is the only report that has gone to the Supreme Court and been accepted in the State of Pennsylvania. That report was objected to by the county authorities and our assessments were raised. It was then appealed to the Supreme Court, and after the Supreme Court stated that the foot-acre method was not legal, it added that there was no evidence in this report of the use of that method, and so confirmed that report. This was the only one in the whole State of Pennsylvania that has ever been confirmed by the Supreme Court; so if any of you ever have occasion in Pennsylvania to value mineral property, do not tell the court how you do it.

S. A. TAYLOR, Pittsburgh, Pa.—An interesting matter came to my attention this week; while it is not exactly in line with this discussion, yet it is a matter that has some bearing on it. It is this: The Internal Revenue Act, providing the tax on incomes, recently passed by the national government, among other things changed the conditions as to the taxing of corporations, especially mines. Prior to this act becoming a law, corporations were permitted to deduct from their net income the

sum of $5,000, and an additional amount equal to that necessary to provide for the entire depreciation of their property.

The new law not only eliminates the flat deduction of $5,000, but also provides that the deduction for depreciation of property cannot exceed 5 per cent. of the gross value or sales price of the product of the mines, at the mines. This does not permit the companies to deduct the whole of the depreciation of their properties in case that depreciation exceeds the 5 per cent. allowed by the law; for example, in this Pittsburgh district the average sales price of coal at the mines is approximately $1.20 per net ton, 5 per cent. of which would be 6c. per ton, the amount permitted to be charged off as depreciation of property, while generally it would cost about 15c. per ton to replace that coal, and in many cases it would be 20c. per ton, which shows at once that this law is unfair to the coal operator of this district to the extent of from 9c. to 15c. per ton. In the Connellsville coke district a very unfair condition is also shown. The royalty on coke alone is approximately 35c. per ton. The fair average sales price for coke this last year would be $2 per ton; 5 per cent. of this equals 10c. per ton with which to replace the 35c., the actual value of the property.

The iron-ore mines of Michigan are more unfairly treated under the provisions of this new act than either the coal or coke companies above referred to. In many cases the royalty for iron ore will run as high as $1 per ton. The sales price of the ore at the mine is about $3 per ton, 5 per cent. of which is 15c., which will be allowed under this law for what costs $1. I understand the iron-ore men have taken the matter up with the government for adjustment. The collectors of internal revenue acknowledge the unfair provisions of the law, but they are helpless to give any relief in the matter. It is thought by some lawyers that the only way this matter can be corrected is to have that part of the law repealed or amended, as the government cannot be sued and it is doubtful whether or not any decision of the Revenue Department can be legally made which will give equitable relief in this matter.

This whole affair shows what an injustice can be perpetrated upon the taxpayers by legislatures with little or no knowledge of conditions pertaining to the subject of their legislation.

R. D. Hall, New York, N. Y.—I think sometimes it is a little unfortunate that we hear about taxation problems almost entirely from the viewpoint of the anthracite region. The problem in that section of the coal-mining industry is largely different from that of the rest of the country. When Mr. Norris and others propose that the tax be put upon output, they do it, I believe, with a due sense of their responsibility to the community, and with the anticipation that under such a plan a fair amount of taxation would be paid by coal-land owners to the various treasuries; but, unfortunately, there are some people in other coal fields of this country who have large holdings and they would keep them for an in-

definite time without use, if absolutely no taxation had to be paid until such time as they extracted the coal. As an example of such a large landholder who is desirous of holding land because he has to pay no taxes, I would bring to your attention "Uncle Sam," who puts a high price on all his holdings, and who is able to maintain that excessive valuation because under the law of the United States, Uncle Sam pays absolutely no tax on what coal lands he possesses. And I think not only he, but also several other smaller individuals, would be not a little disposed to hold on to their land indefinitely if you taxed their coal only on the output and they were able to hold it without any charge at all for taxation. It would be manifestly unjust to those people who are developing and working their coal and to those other citizens who are living in that particular district, whose property is more or less depreciated by the fact that the resources are not being used, if that man were allowed to hold his property without taxation until such time as he chose to develop it.

Mr. Griffith.—That may all be regulated by law, as in the case of unseated land at present. If a man owns a good farm and a portion of unseated land, he is not taxed on the latter the same as on the good land; assessors simply put a nominal value on the unimproved land, sufficient to let the owner know that the State appreciates that he has land there. To remind him that that part of his land may be of more value later they tax it every year, but not as much as they do his fine farm land. So with coal lands and mineral lands of any kind which are not being developed, there should be a nominal tax placed, sufficient to prevent holding it indefinitely for speculative purposes, but not enough to make it a burden to hold for future mining.

Mr. Evans.—Did I understand you to say that royalty could not be charged against operating expense?

Mr. Taylor.—If you are paying royalty, you are permitted to charge off the royalty as the cost of mining. But if you unfortunately own the coal, you cannot charge it off. The government permits the royalty to be charged off as part of the cost of mining, and yet if you own the coal you will only be permitted to take off 5 per cent. of the sales price of the coal at the mine mouth.

The Pittsburgh Coal Field in Western Pennsylvania

BY H. A. KUHN, PITTSBURGH, PA.

(Pittsburgh Meeting, October, 1914)

THE Pittsburgh coal field in western Pennsylvania is conceded to be the most important in the world. To measure its importance it is necessary to understand the extent of its service in the various industries of the country. Probably 90 per cent. of the pig iron manufactured in the United States up to the present time has been made by using coke manufactured from the Pittsburgh coal seam in western Pennsylvania. This coal field is the foundation on which the city of Pittsburgh rests and is the reason for the great growth of the iron industry in the Pittsburgh district. Iron ore is brought to this district, not because Pittsburgh is a natural location, over other locations, for the iron and steel industry—as its position places the manufacturer at the mercy of the railroads—but the ore is brought 1,100 miles to meet the fuel. It can be said that the illuminating-gas industry in the United States has used this coal exclusively to the same extent that the pig-iron maker has used it. It may be also said that 20 to 25 per cent. of the fuel used on railroads in the United States comes from this coal field.

The Pittsburgh coal field is unquestionably the center of the industrial population of the United States, for in addition to the industries of the district and those closely adjoining, it has tributary to it all the cities and industries along the Great Lakes and practically all of Canada with the exception of the extreme western and eastern ends. It supplies the industries and population west of Duluth and Superior many hundreds of miles. This coal is floated down the Ohio and Mississippi rivers, supplying the towns *en route*, and is delivered in New Orleans, a distance of 2,200 miles, for approximately 80c. to 90c. per ton transportation cost. It is delivered on the docks of Superior and Duluth at a cost of transportation 50c. a ton less than the cost of transporting the same coal from Pittsburgh to a local consumer in Philadelphia. With other Appalachian coals it has large markets east and along the seaboard especially for by-product-coke making and the illuminating-gas industry. It is considered the premier railroad fuel of the world on account of the fact that this coal in a given-sized locomotive, will probably haul more cars than any other coal in the world. Tests made at Altoona by the motive-power department of the Pennsylvania Railroad Co. show that Pittsburgh gas coal

evaporates as high as 18.9 lb. of water per square foot of heating surface. It is stated that the lower volatile coals, with a theoretically higher heat value per pound of fuel, do not evaporate more than 12 or 13 lb. of water per square foot of heating surface. For this reason this important railroad has adopted this coal as its standard fuel. By its use with the same crew and engine a maximum number of cars may be hauled.

These statements explain somewhat the reason for the enormous production of coal from this field and seam.

The production of coal from the Pittsburgh seam in western Pennsylvania in 1913 was in round figures approximately 100,000,000 tons; in Ohio from the same seam of coal approximately 20,000,000 tons; in West Virginia 10,000,000 tons, making a total tonnage of approximately 130,000,000 out of a total production of 470,000,000 tons of bituminous coal produced in the United States in the same year. Of the 170,000,000 tons of bituminous coal produced in Pennsylvania approximately 100,000,000 tons comes from the Pittsburgh seam. Fayette and Westmoreland counties with only a small acreage left of unmined coal are now producing approximately 60,000,000 tons of coal annually; Allegheny and Washington counties produce about 20,000,000 tons each; Greene county has only recently started as a producer of coal but a capacity is now established in this county which is upward of 2,000,000 tons a year.

The geology of the Pittsburgh coal measure is so well understood that it is not worth while discussing this field from that standpoint. Some practical features, however, which relate to its geology, are interesting. The main feature of this seam from a geological standpoint is its persistency and integrity over the area which it covers. Whether the coal has been opened or not, those who are familiar with the coal in each zone or subdistrict (and the slow but uniform change from one basin to the next) can state unreservedly how thick the coal is, how much sulphur it will probably contain within a small percentage and about how many tons per acre may be recovered. Thousands of miles of entries have been driven through this coal field in mining operations of the district and thousands of oil wells and gas wells have been drilled through it where it is not being mined. Several facts touching the deposit of this coal seam are worth mentioning. It is evident from an inspection of the geological maps of Pennsylvania, Maryland, and West Virginia that the plant life which formed this field of coal was extremely luxuriant, increasing in depth or density toward the east, and that the purest part and largest part of this coal field has been lost through erosion. This seam in Georges Creek field near Cumberland, Md., measures as much as 16 ft.; in its western outcrop in Ohio about $3\frac{1}{2}$ to 4 ft.; in the northwestern end of Allegheny county in western Pennsylvania near the State line about $4\frac{1}{2}$ ft. of the main or working bench of coal and $2\frac{1}{2}$ to 3 ft. of roof coal

measures, the working bed being separated from the roof measures by what is known as the "draw slate" of the district, which averages from 8 to 14 in. In the Connellsville field the coal at places attains a thickness of 8 ft. and on the southern boundary of Greene county near the Monongahela river it measures 9 to 10 ft. in thickness. In the southern extension of this seam in West Virginia the roof coal and main or working bed of the Pittsburgh district become practically one solid bench of coal with only thin laminations of slate marking the divisions which are so noticeable in the northern part of the field.

It may be said from an inspection of the geological map and from a knowledge of the quality of the coal that the plant growth that formed this coal was much more luxuriant in the eastern end and that the coal was deposited with less impurities and under quieter conditions. The coal thins going west, but the extreme purity of the coal extends westward to the Pinhook anticlinal running southwest from Pittsburgh through Allegheny and Washington counties; and west of the Pinhook anticlinal the change is gradual, the coal becoming more impure, higher in ash and sulphur with the distance westward, and the working bed or bench becomes thinner. The coal containing the least impurities from this seam in the Pittsburgh district is mined from what is known as the "Gas Coal Basin," which lies between the Pinhook anticlinal and the Murraysville anticlinal on the west and the Waynesburg and Saltsburg anticlinal on the east. The working bed of coal in this basin averages about 6 ft. thick. Above this bed and separated from it by the draw slate, 2 to 4 ft. of roof coal is deposited. The main working bed in this basin is prepared for market containing about 6 per cent. non-combustible matter, and in some cases less than this, and the average coal contains not over 1 per cent. sulphur. Coal in the Connellsville basin is equally pure, but on account of the fact that more of the seam is mined in this field the ash is usually higher than in the coal produced in the gas-coal field, the sulphur being about the same. In recent years it has been discovered that the lower or southwestern portion of the gas-coal zone makes a standard coke, the difference between it and the Connellsville coke being that the coal in this basin on account of its hardness has to be crushed before it is put into the coke ovens, or when not crushed only the slack coal is used. This hardness of the coal in the southwestern portion of the gas-coal basin, however, is an advantage on account of the fact that the lump coal from the gas-coal basin is sold at the mine for shipment at a normal price of $1.40 per ton, which is equivalent to coke (including its cost of manufacture based on beehive oven practice) at $2.50 per ton at the ovens.

The Pittsburgh coal field in western Pennsylvania west of the Monongahela river and within the State lines comprises about 850,000 acres of unmined coal today. In order to understand the available tonnage per acre in this field of coal and the values in various parts of the coal field

it is necessary to subdivide it into zones of quality and thickness; and in order to understand the value from all standpoints it is necessary to further subdivide it into zones of thickness, zones of quality, zones of cost with reference to the cost of mining and cost of transportation. The thickness of the coal in the various parts of the field determines the available tons per acre which in turn becomes a factor in the valuation of the land and the cost of mining. To measure the value of the land of the different subdivisions or zones all of the factors, quality, tons of recoverable coal to the acre, the cost of mining and the cost of transportation to market must be considered.

The author in considering this coal field finds it more interesting from the utilitarian point of view, for after all that is the true test of all fuel resources. Its past and present service to the industries of the country, the future uses to which it may be put, its conservation, its probable life, now susceptible of definite measurement, are the principal factors which give interest to its consideration. The value and virtue of the different subdistricts unfold from a study of them under these subjects. Another practical feature that to-day attracts attention to this or any other coal field is the railroad tariffs for the transportation of the product, which will from the present time be adjusted by the Interstate Commerce Commission from the standpoint of cost and service of the railroads, the welfare of the public being properly considered.

To measure the life of this coal field is an easy matter. In the Connellsville basin, the Lower Connellsville or Klondike basin, and other areas of coal east of the Monongahela river there are approximately 100,000 acres of the Pittsburgh seam available and unmined at this time. In 1912, I determined the unmined acreage of coal in the Connellsville basin and also the Lower Connellsville basin and the ovens attached to the lands by copying from the assessors' books in each township of these two fields the sworn statements of the various owners of the coal lands and coke plants. From these sworn statements I made the following summaries:

Connellsville Field. Ovens and Acreage in 1912

	Number of Ovens		Acres
	Tax List	Connellsville Courier	
Old Basin	22,985	20,361	32,461
Klondike	12,784	12,377	33,029
Total	35,769	32,738	65,490

Connellsville Field. Ovens going Out of Blast Due to Exhausted Coal Acreage

	Old Basin	Klondike	Total
In next 3 years	5,393	1,310	6,703
In next 5 years	7,097	2,052	9,139
In next 10 years	11,940	3,814	15,744
In next 15 years	14,529	5,270	19,789
In next 17 years	17,491	6,130	23,621

After 17 years there will be, of all ovens shown, 5,494 ovens in the Old Basin with 5,881 acres of coal left, with an average life of over 9 years; and 6,654 ovens in the Klondike with 9,501 acres of coal left with an average life of over 13 years.

Connellsville Field. Ovens going Out of Blast Due to Exhausted Acreage
(Eliminating the H. C. Frick Coke Co.)

Merchants Ovens

	Old Basin	Klondike	Total
In next 3 years	2,575	1,310	3,885
In next 5 years	3,135	2,052	5,187
In next 10 years	4,691	3,814	8,505
In next 15 years	6,455	4,760	11,215

After 15 years there will be of the present oven plants shown 1,039 ovens in the Old Basin, with 1,653 acres of coal left, with an average life of over 14 years; and 3,842 ovens in the Klondike with 5,781 acres of coal left, with an average life of over 13 years.

Number and Life of Ovens of H. C. Frick Coke Co. in Connellsville Field

	Number of Ovens			Average Life, Years
	Tax List	Connellsville Courier	Acres	
Old Basin	15,491	14,199	23,896	14+
Klondike	4,182	3,664	14,271	31+

In addition to the above the Frick Coke Co. has 2,935 acres in Redstone township, 1,043 acres in Menallen township and 4,423 acres in Luzerne

township on which there are apparently no ovens. This coal together with considerable acreage in adjoining townships will probably be shipped to by-product ovens, making the total life of the Steel Corporation's coal lands in the Klondike district approximately 20 to 25 years, assuming that they draw on these lands to make up the decline in the Old Basin.

It is clear from this accurate information that the original coking coal fields of western Pennsylvania comprising these two basins of Pittsburgh coal seam, which were thought to be inexhaustible a few years ago, have only a short life. More astounding than this, however, is the short life of what is known as the Youghiogheny or Westmoreland gas coal, which comprises the coal in the gas-coal basin extending from Irwin on the main line of the Pennsylvania Railroad, east of Pittsburgh, in a southwest direction, crossing the Youghiogheny and Monongahela rivers and extending into the northern end of Greene county. The coal from this basin has supplied probably 90 per cent. of the coal-gas illuminating plants in the United States up to the present time. With the disuse of oil in gas making, steel heating and melting and in many other industries, the demand for this grade of coal will probably double within the next five years. Including the Westmoreland field and extending into the northern end of Greene county there is unmined and available in this basin approximately 150,000 acres. On account of the fact that the southwestern portion of this basin makes a standard low-sulphur coke, various steel corporations have purchased large bodies of this land and there is now only available for the open market the product from about 70,000 acres of this coal. The short life of the anthracite field is usually commented upon, but it can be stated that the commercial coal in this particular field and grade of coal will be exhausted in one-seventh of the time necessary for the exhaustion of the anthracite coal, assuming that the peak of production in the anthracite field has been reached. The basin of coal lying west and northwest of this basin between the Pinhook and Washington anticlinals will be called upon to supply the gas coal from the Pittsburgh district when the gas-coal basin proper has been exhausted. In this discussion, however, all lands west of the Pinhook and Murraysville anticlinals are included under the name of "Panhandle" coal.

Fig. 1 shows the number of acres unmined in each subdistrict, the number of tons of recoverable coal per acre based on the past practice of the district in mining, and the divisions of quality in the district. Fig. 2 shows the divisions in the cost of mining and the freight-rate divisions, all of which go toward making the values of the various lands of the district. Fig. 3 shows the estimated life of the various divisions of the Pittsburgh coal field. The coking-coal lands are divided into those available and unmined containing 1 per cent. sulphur and under. This

includes the Connellsville basin, the greater part of the Klondike field and the eastern portion of Washington county east of the Pinhook anticlinal and southwest of Monongahela City, and the field in the northeastern portion of Greene county. The coking coal higher than 1 per cent. in sulphur and which must be washed in order to make standard coke includes the upper portion of the Connellsville basin, the southern portion of the Klondike field and the southeastern portion of Greene county. The extreme thickness of the coal and large recovery of coal

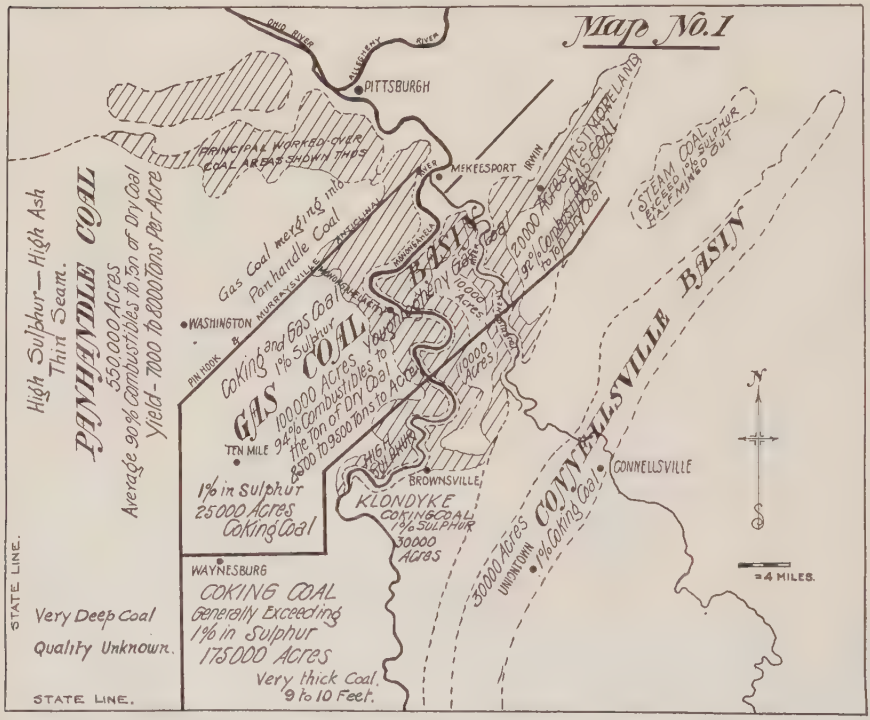


FIG. 1.—SHOWING NUMBER OF ACRES UNMINED AND NUMBER OF TONS OF RECOVERABLE COAL PER ACRE. ALSO DIVISIONS OF QUALITY.

to the acre in the southeastern portion of Greene county and the low mining cost offset the cost of washing.

In calculating the life of the various fields of coking coal, the pig-iron production tributary to this field of coal up to October, 1912, as given by the *Iron Age*, with a forecast showing the normal increase, is used as a basis. It is also assumed that the new fields will replace the declining production of the Connellsville basin and will also take on the burden of the increased consumption due to increased production of pig iron geographically tributary to these coal fields.

The average flat rate of increase of pig-iron production, based on figures for 1900 to 1910, was approximately 1,171,334 tons. For the

next 10 years 1,400,000 tons may be assumed as the average flat rate of annual increase. The productions from 1912 to 1920 would, therefore, be as shown below. This forecast allows little for increase in population or increased use of iron per capita.

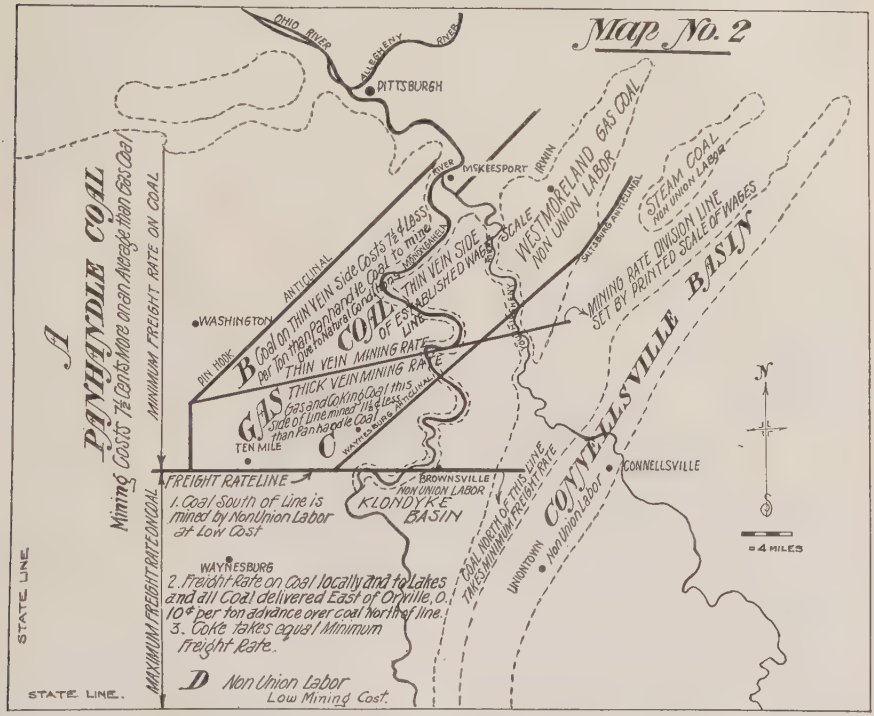

Map No. 2
OHIO RIVER
ALLEGHENY RIVER
PITTSBURGH
MONONGAHELA RIVER
McKEESPORT
IRWIN
WESTMORELAND GAS COAL
NON UNION LABOR
STEAM COAL NON UNION LABOR
SMITHFIELD
MINING RATE DIVISION LINE SET BY PRINTED SCALE OF WAGES
CONNELLSVILLE BASIN
A PANHANDLE COAL
Mining Costs 7½ Cents More on an Average than Gas Coal
MINIMUM FREIGHT RATE ON COAL
WASHINGTON
ANTICLINAL
B Coal on THIN VEIN Side Costs 7½ ¢ (Less) per Ton than Panhandle Coal to mine
Due to Natural Conditions
COAL
OF ESTABLISHED WAGES
THIN VEIN SIDE
THIN VEIN MINING RATE
THICK VEIN MINING RATE
GAS
Gas and Coking Coal this side of Line mined 11½ ¢ less than Panhandle Coal
C
TEN MILE
WAYNESBURG
FREIGHT RATELINE
BROWNSVILLE
NON UNION LABOR
KLONDYKE BASIN
1. Coal South of Line is mined by Non Union Labor at Low Cost
2. Freight Rate on Coal locally and to Lakes and all Coal delivered East of Orrville, O. 10¢ per ton advance over coal North of line.
3. Coke takes equal Minimum Freight Rate.
D Non Union Labor Low Mining Cost.
COAL NORTH OF THIS LINE TAKES MINIMUM FREIGHT RATE
UNIONTOWN
Non Union Labor
CONNELLSVILLE
N
= 4 MILES
STATE LINE
MAXIMUM FREIGHT RATE ON COAL
STATE LINE.

FIG. 2.—DIVISIONS IN COST OF MINING AND FREIGHT RATES.

Year	Average Pig Iron Production Tons	Coke Required 1.56 Tons Coke to 1 Ton Pig Iron Tons
1912	29,115,000	45,419,000
1913	30,515,000	47,603,000
1914	31,915,000	49,787,000
1915	33,315,000	51,971,000
1916	34,715,000	54,155,000
1917	36,115,000	56,339,000
1918	37,515,000	58,523,000
1919	38,915,000	60,707,000
1920	40,315,000	62,891,000

This following statement shows: 1. That 75 to 80 per cent. of all the iron production of the United States is located tributary to the coking coal of the Pittsburgh district.

2. The quantity of coke required by the different iron-producing districts. This is arrived at by taking the pig-iron production of the United

States for the last five years and the coke production for a like period. The pig-iron production of the United States from 1907 to 1911, inclusive, was 117,445,261 tons, and the coke production from 1907 to 1911, inclusive, was 183,388,466 tons. Coke required (for blast furnaces, foundries, smelters, and gas production) in terms of 1 ton of pig-iron production equals 1.56+ tons of coke production.

District	No. of Stacks	Month of October, 1912. Pig-Iron Production, Tons		Coke Required, Tons	
Buffalo, New York	16	179,726		280,372	
Rest of New York	2				
Pittsburgh district	51	621,813		970,028	
Eastern Pennsylvania (small stacks)	35	227,106		354,285	
Western Pennsylvania	18	157,027		244,962	
Shenango valley (Penn.)	16	143,115		223,259	
Mahoning (Penn.)	20	256,711		400,469	
Central and Northern Ohio	21	237,506		370,509	
Wheeling district (Ohio and W. Va.)	10	118,037		184,137	
	189		1,941,041		3,028,012
Chicago district (Mich., Wis. and Minn.)	38	442,091		689,661	
	38		442,091		689,661
Southern Ohio	9	38,419		59,933	
New Jersey	1	5,370		8,377	
Southern States	41	239,686		373,910	
Western States	3	23,326		36,390	
	54		306,801		478,610
	281		2,689,933		4,196,292

This analysis shows that the net demand on the Pittsburgh district by the different iron-producing districts, is approximately 35,000,000 tons of coke per year. This figure is arrived at by taking the iron production for October, 1912, of the States of Pennsylvania, New York, and northern Ohio, with one-half that of the Chicago and the northern Lake district. These requirements show an annual demand for 40,485,212 tons of coke, from which must be subtracted 4,136,000 tons of coke manufactured in by-product coke ovens in Pennsylvania, New York, and Ohio, and also subtracting 1,000,000 tons of hard coal used in iron smelting in eastern Pennsylvania.

Using the foregoing forecast as a basis and assuming that the same percentage of pig-iron production tributary to the Pittsburgh coal field will be maintained out of the whole production of the country, the life of the various subdistricts of coking coal may be stated approximately as follows: Connellsville basin, 15 years; Lower Connellsville or Klon-

dike, 20 years; eastern Washington county, 25 years, it being assumed that one-half of this coal will be used for smelting iron ore and the other half used for gas making, steel melting and other purposes; 175,000 acres in eastern Greene county, 40 years.

Merging all the acreage together the life of the coking coal in the Pittsburgh district based on a normal increase in the use of this coal in the iron and steel industry and as a domestic fuel to replace hard coal will be approximately 40 years. It may be assumed that the present type of ovens will be largely replaced by by-product oven plants on the field and at the point of manufacture.

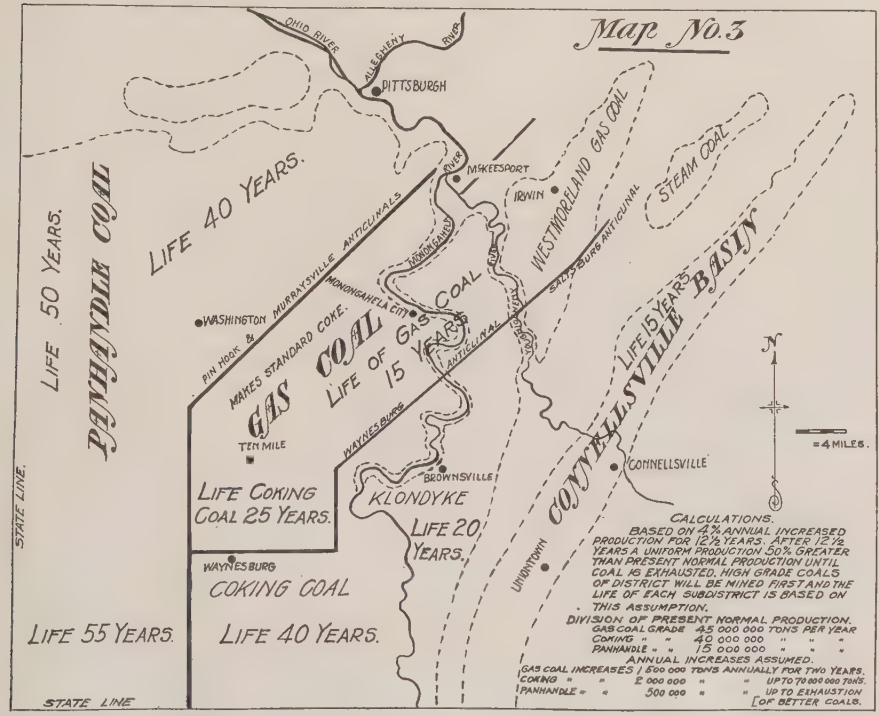

Map No.3
OHIO RIVER
ALLEGHENY RIVER
PITTSBURGH
McKEESPORT
IRWIN
WESTMORELAND GAS COAL
STEAM COAL
LIFE 40 YEARS.
LIFE 50 YEARS.
PANHANDLE COAL
MONONGAHELA RIVER
MONONGAHELA ANTICLINAL
WASHINGTON
MURRAYSVILLE ANTICLINAL
PIN HOOK &
MAKES STANDARD COKE.
MONONGAHELA CITY
GAS COAL
LIFE OF GAS 15 YEARS.
ANTICLINAL
YOUGHIOGHENY RIVER
SALTSBURG ANTICLINAL
LIFE 15 YEARS.
CONNELLSVILLE BASIN
GAS
TEN MILE
WAYNESBURG
BROWNSVILLE
CONNELLSVILLE
=4 MILES.
LIFE COKING COAL 25 YEARS.
KLONDYKE
LIFE 20 YEARS.
UNIONTOWN
WAYNESBURG
COKING COAL
LIFE 55 YEARS.
LIFE 40 YEARS.
STATE LINE.
STATE LINE
CALCULATIONS.
BASED ON 4% ANNUAL INCREASED PRODUCTION FOR 12½ YEARS. AFTER 12½ YEARS A UNIFORM PRODUCTION 50% GREATER THAN PRESENT NORMAL PRODUCTION UNTIL COAL IS EXHAUSTED. HIGH GRADE COALS OF DISTRICT WILL BE MINED FIRST AND THE LIFE OF EACH SUBDISTRICT IS BASED ON THIS ASSUMPTION.
DIVISION OF PRESENT NORMAL PRODUCTION.
GAS COAL GRADE 45 000 000 TONS PER YEAR
COKING " 40 000 000 " " "
PANHANDLE " " 15 000 000 " " "
ANNUAL INCREASES ASSUMED.
GAS COAL INCREASES 1 500 000 TONS ANNUALLY FOR TWO YEARS.
COKING " " 2 000 000 " " UP TO 70 000 000 TONS.
PANHANDLE " " 500 000 " " UP TO EXHAUSTION OF BETTER COALS.

FIG. 3.—ESTIMATED LIFE OF VARIOUS DIVISIONS OF PITTSBURGH COAL FIELD.

The conditions governing the value of the Pittsburgh coal field both by the ton and by the acre may be stated as follows:

The Pittsburgh coal field has been so largely operated and the coals from the different subdistricts tested that it is not only possible to determine approximately the number of tons of recoverable coal per acre but it is possible to determine the value of the land based on the number of heat units delivered per pound of coal. The pure gas coal in the gas-coal basin as compared with the average Panhandle coal as a steam-generating fuel (the noncombustible matter including ash, sulphur and water) may be stated as follows: The purest gas coal is prepared for

market with about 94 per cent. combustible matter and 6 per cent. noncombustible matter, the Panhandle coal is prepared for market with an average of 90 per cent. combustible matter and 10 per cent. noncombustible matter, the average difference being 4 per cent. The average delivered cost of a ton of Pittsburgh coal as it is now delivered to consumers, including freight rates, approximates $2.50 per ton. (The lowest delivered cost in the Pittsburgh district is about $1.50 per ton. Coal shipped by way of the Great Lakes and into Canada and the West is delivered to the consumer at from $3.50 to $4.50 per ton, the average, giving due weight to the tonnage, is approximately $2.50 per ton.) A difference of 4 per cent. in heat value on a basis of $2.50 delivered cost per ton shows a difference in delivered money value of 10c. per ton in favor of gas coal, and reduced to an acreage basis for gas coal amounts to an excess value of between $800 and $900 an acre. As has been stated before, the gas coal merges gradually from the Pinhook anticlinal westward into Panhandle coal and there are points and locations immediately west of the gas-coal basin where the difference in value of heat units per pound of coal delivered would be much less than the average difference.

The value of the coal in the Pittsburgh district based on quality as applied to any special industry may be reduced to a value per acre of land. It has been stated by managers of illuminating-gas plants that the Pittsburgh gas-coal product has an ultimate excess value of 25c. to 30c. per ton of coal over gas coals from other districts. The coking coal of the Pittsburgh seam in western Pennsylvania, based on physical and chemical properties and geographical location and freight rates, is given possession of approximately 80 per cent. of the coke demand of the United States and Canada. Judge Gary, of the United States Steel Corporation, is credited with a public statement placing $2,000 an acre as the value of coking coal from the Pittsburgh seam in western Pennsylvania. This checks up with a statement made to me by one of the Pig Iron Committee of a large steel corporation that a coke with the best physical structure containing 1 per cent. or less of sulphur is worth about 30c. a ton more in the manufacture of pig iron as compared with higher sulphur coke, as additional coke and flux must be placed in the furnace in order to remove the excess sulphur.

The geographical value of the Pittsburgh coking coal based on freight rates to distant markets has not been fully grasped either by the steel companies or the public. For by-product-coke making or gas making along the Great Lakes, at any point, it is possible to ship low-sulphur Pittsburgh coking coal to the lake front for a delivered freight cost placed on board vessel of 83c. a ton, the vessel rate from Lake Erie ports to any point on the Great Lakes ranges from 25c. to 35c. a ton, making a total freight rate to Lake Superior points of $1.13 and to Lake Michigan points $1.18 per ton. The minimum railroad freight rate on coal from West

Virginia and Kentucky to Chicago or Lake Michigan points is $1.90 per ton. A freight rate from southern Illinois based on about 3 mills per ton-mile would be $1.05 per ton to Gary, Ind. Coal shipped from the Pittsburgh coal field via lake to Lake Michigan points therefore shuts out all West Virginia and Kentucky coals by rail and can be put into the storage yard at the by-product coke-oven plant in the Chicago district along the lake front (based on available carbon per unit of pig iron produced) at about an equal cost with southern Illinois coal. Southern Illinois coal, such as Franklin County coal, according to a paper on Illinois coals,[1] contains from 18 to 21 per cent. noncombustible matter as against 6 to 7 per cent. noncombustible matter in Pittsburgh by-product coal. I believe that recent developments in southern Illinois indicate that these analyses of southern Illinois coals are too high in noncombustible matter, but taking this coal as now produced in the railroad car, 16 per cent. may be assumed as noncombustible matter, divided approximately into 9 per cent. ash and 7 per cent. water, or 10 per cent. ash and 6 per cent. water. There would, of course, be certain losses in heat units in the coking of southern Illinois coal on account of the water that could not be charged to Pittsburgh coal, and excess sulphur in Illinois coal would also reduce the amount of available carbon per unit in the blast furnace. The foregoing facts when taken in connection with the superior coking qualities of Pittsburgh coking coal would indicate that southern Illinois coal for by-product use in the Chicago district would not be as cheap ultimately as Pittsburgh coal shipped there by lake, based on equal cost of coal at the mines. A careful study of the situation indicates also that the pig-iron furnaces bordering on the Great Lakes, almost all the furnaces in northern and central Ohio, western Pennsylvania, furnaces in New York State, a large percentage of the furnaces in eastern Pennsylvania, and the furnaces in Canada will draw their fuel supply from the Pittsburgh coal field; as that industry increases in volume, the coal production will have to increase in the Pittsburgh coking-coal field and the life of the field will be shortened accordingly.

Another factor that governs the value and life of the subdistricts in the Pittsburgh coal field is involved in the cost of mining (see Fig. 2). The wage scale divides the field into two districts (known locally as the thin- and thick-vein mining-scale districts), the dividing line running from Port Royal on the Youghiogheny river west through Lock 4 on the Monongahela river, and extending still farther west through a point north of Bentleyville, Washington county. The coal north of this line has the thin-vein wage scale, which reduced on the face of the scale to a run-of-mine basis amounts to about 7c. per ton excess cost for cutting and loading, as compared with coal south of this line. This, however, is not a true measure of the cost of mining coal in this district, as natural

[1] *Trans.*, xl, 4 *et seq.* (1909).

conditions in various parts of the field modify the cost of the coal as it is finally loaded for market.

The Pittsburgh Coal Co. made sworn statements before the Interstate Commerce Commission at Washington on the cost of mining 18,000,000 tons of coal from Apr. 1, 1910, to Aug. 31, 1911, as follows:

	Production, Tons	Average Cost Mine Run Coal. (Does not include interest or dividends)
Panhandle coal	8,174,880	$1.0434
Gas coal	5,070,600	0.9686
Thick-vein coal (coal south of wage-scale line).	5,052,887	0.9283

This statement, on account of the fact that the tonnage is extremely large and covers a long period of time, and the further fact that the same accounting system was applied to the whole tonnage, and in like manner to each district, makes it an accurate measure of the cost of mining Pittsburgh coal in the various subdistricts of the field, and for purposes of current operation places an accurate value on the land in the different districts. From the standpoint of cost it will be seen that the gas coal marked area *B* in Fig. 2 was produced on board railroad cars for practically 7½c. per ton less than Panhandle coal. Reduced to an acreage basis this indicates that for current operating purposes gas coal costs $600 to $700 an acre less to mine than Panhandle coal. As a steam-generating fuel it has been noted before that it is actually worth, at a delivered cost price of $2.50 a ton, 10c. per ton more to the consumer, which makes its excess value above that of the average Panhandle coal about $1,400 an acre, assuming that the consumer is willing to pay for the coal based on the heat units delivered. The thick-vein coal was mined for 11½c. a ton less than the Panhandle coal, or when reduced to an acreage basis was mined for $1,000 an acre less than Panhandle coal. As a large part of the thick-vein coal comes under the head of coking coal and gas coal, its value, based on delivered results, is almost equal to the value placed on coking coal by Judge Gary, as compared with average Panhandle coal, assuming the average Panhandle coal value at $200 to $300 per acre.

Another light may be thrown on the value of Pittsburgh coal lands by the sworn testimony of the Pittsburgh Coal Co. before the Interstate Commerce Commission involving the same period of time, by combining the foregoing statement with Comptroller Hornberger's Exhibit No. 5. Exhibit No. 5 gives an average market value of all grades of coal at the mine for lake shipment during 1910 of $1.0708 for run-of-mine coal. For purposes of calculation it may be assumed that all three grades of coal were sold at the same price, and based on this assumption, the 8,174,800 tons of Panhandle coal earned the Pittsburgh Coal Co. $224,000, the 10,123,487 tons of gas coal and thick-vein gas and coking coals

earned this company approximately $1,338,000, provided the low-grade Panhandle coal was sold at equal prices with the higher-quality coal, otherwise all the profits must have been made out of gas coal and the adjoining thick-vein gas and coking coal.

With all these accurate data in hand it is very easy to place values on the different subdistricts of the Pittsburgh coal field when measured by governing conditions within the district itself. When compared with other coal fields and districts in West Virginia and Kentucky it is found that the excess freight rate now charged on coal from the Fairmont district in West Virginia in the same seam of coal to the lake front is 12 c. per ton. As compared with Kanawha coals, southern West Virginia and Kentucky coals, the excess freight rate to the lake front paid by these coals amounts to 19c. per ton and upward. As compared with all-rail shipments as far west as Chicago the freight rate on West Virginia and Kentucky coals lying nearest to Chicago is the same as the Pittsburgh rate, but by way of the lake the Pittsburgh field has an advantage of 12c. to 19c. per ton over these coals together with a higher market value based on delivered results. The tendency of freight-rate adjustment will add additional value to the Pittsburgh coal field in western Pennsylvania, as is illustrated by the attempt of the C., H. & D. R. R. to haul coal to the lake front for approximately 2 mills per ton-mile as compared with 5 mills per ton-mile paid by Pittsburgh coal. It is reasonable to believe that changes in freight rates from the Pittsburgh district will be in reductions and the changes in freight rates from southern West Virginia and Kentucky will be necessary increases sufficient to maintain the solvency of the railroads carrying the coal.

The Interstate Commerce Commission will find it necessary to fix freight rates more nearly on the basis of cost of transportation in order to maintain the solvency of the railroads themselves and to conserve the natural resources of the country, all of which will tend to conserve the Pittsburgh coal field and probably extend its life over the estimates made. With the exception of the Connellsville field it may be stated that the Pittsburgh coal areas worked over, up to the present time, have not shown a recovery of more than 60 or 65 per cent. In the old days of mining only the breast coal was mined and only cheaply mined portions of this bench were hauled to the tipple. In recent years the more modern operations in the Pittsburgh district have adopted systems of extraction which permit a recovery of 85 per cent. and in some cases 90 per cent. may be secured, but this percentage of recovery only represents the last few years and only a portion of the district. The destruction of the Pittsburgh coal seam in past mining operations in this district was compulsory on account of the fact that the railroads of the district placed an extremely high freight rate on coal shipped, in order to divide the market with

southern railroads, the result being that with union labor, in combination with extremely high freight rates per ton-mile from the Pittsburgh district, and extremely low freight rates from West Virginia and Kentucky, and non-union labor, it has been possible for operators in the southern fields to enter markets, to a limited extent, which naturally belong to the Pittsburgh coal field. It is obvious that it is bad public policy to permit the issue of railroad securities for transportation equipment, the effect of which is to destroy natural resources in order to secure an unnatural coal market for a distant coal field, and to increase the gross business of a railroad at no profit to the stockholders. As stated before, the present condition of the C., H. & D. R. R. illustrates the fallacy of this policy. The resources of the Pittsburgh coal field and those of the United States cannot be conserved unless the cost of service on railroads is a measure of freight rate. With gradual adjustments made in freight rates to cure this evil it may be assumed that 90 to 94 per cent. of the Pittsburgh coal measures may be recovered. The Connellsville field proper, under the supervision of Thomas Lynch, President of the Frick Coke Co., has employed methods in mining which are the most efficient in the United States. The recovery of coal in the Connellsville field through the system established by Mr. Lynch is about 93 to 94 per cent. The operations in this field from the standpoint of percentage of recovery, safety measures provided for guarding against death and accidents to miners, and living conditions around the mines, are an example for all coal-mining operations of the country. If the remainder of the Pittsburgh coal field through adjustment in freight rates and mining costs, would adopt a modified system of the Connellsville plan of extraction, the life of the Pittsburgh coal field would be increased materially and the wealth of the district proportionately increased.

The author of this paper has over a period of 10 or 15 years developed a system of mechanical mining to be applied to a modified system of extraction as practiced in the Connellsville field by the Frick Coke Co. (Figs. 4 to 5). After spending much time and considerable money in this period of time machines and methods were produced which have apparently reached a stage of perfection. Several of these machines have operated continuously for many months in the Pittsburgh coal field with small repairs and without any delays in operation. The results are so convincing, that it can be announced without reservation that mechanical mining is established and that by means of machines now perfected and operating daily, and by adapting the modified plan of extraction equivalent to the methods in the Connellsville field, the following results may be obtained:

1. That 90 to 95 per cent. of the coal measure may be recovered mechanically.

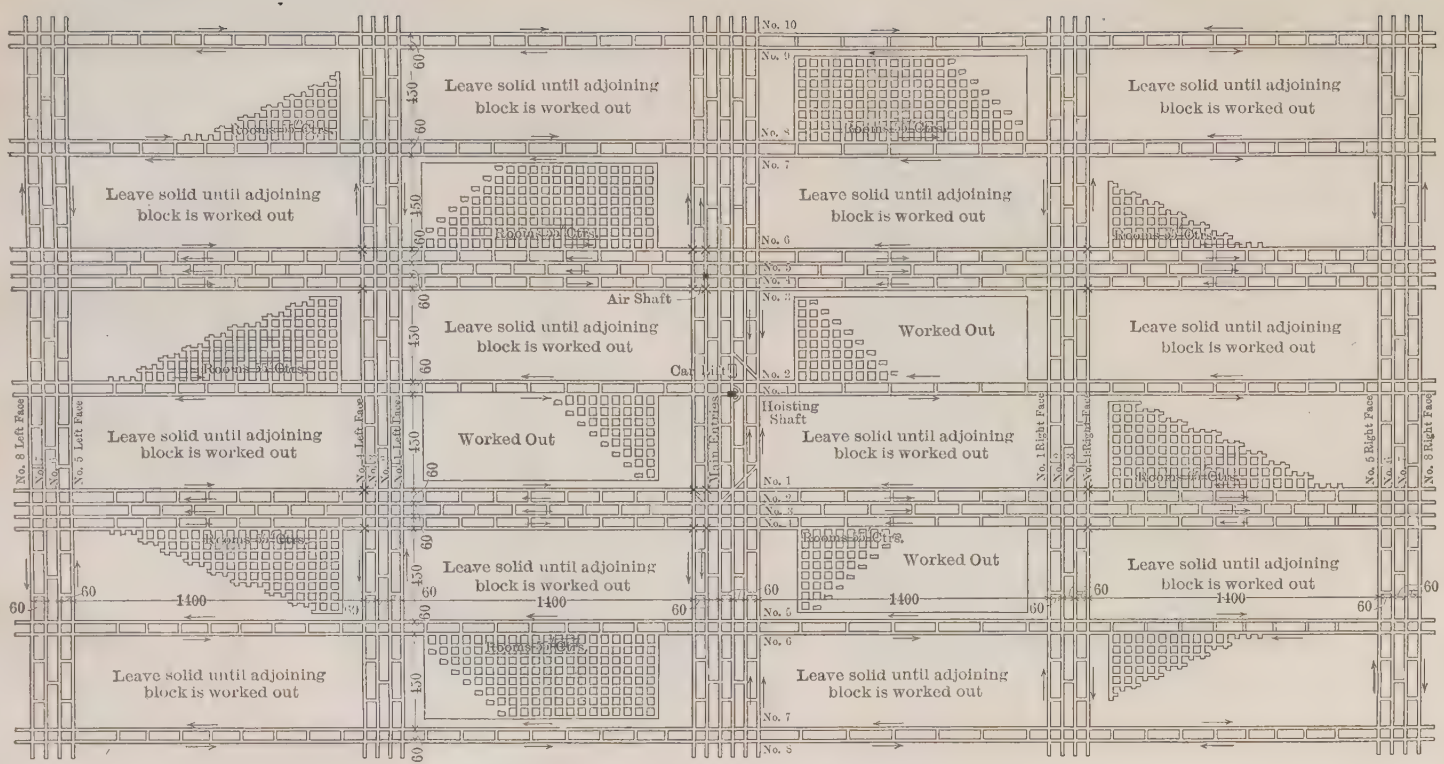


FIG. 4.—PLAN OF MINING, SHOWING EXTRACTION OF COAL BY MECHANICAL MEANS.

2. That the cost of mining in thick seams averaging over 5 ft. will be reduced from one-third to one-half.

3. That one man operating the new machines, and using the methods now perfected and working, will be able to produce as much coal as four to six men produce under the present methods by the use of the present machinery.

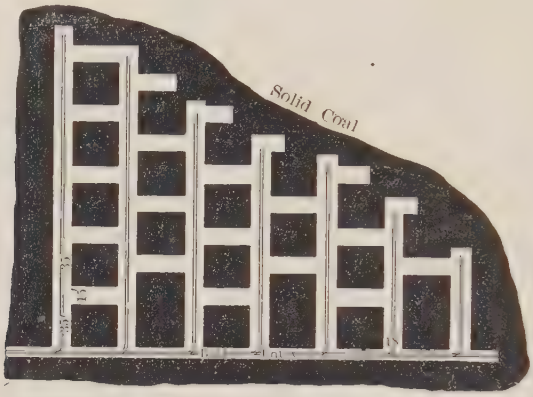


FIG. 5.—PLAN SHOWING ADVANCE.

FIG. 6.—PLAN SHOWING RETREAT.

With all this in view the future of the Pittsburgh coal field is extremely bright and the wealth of the district is increased largely.

It may be stated that the present beehive ovens or open ovens of the district will be used to exhaust areas of coal lands attached, but that future coke plants supplying the iron and steel industry tributary to this coal field will, to a large extent, be the retort type of oven with by-product

recovery. Some of these plants will be placed at the point where the iron and steel is manufactured and other plants will be placed in the coal field. The Pittsburgh district is facing in a few years the total exhaustion of natural gas, or at least a very large reduction in the supply. There are established throughout the district, and crossing the coal field, many large pipe lines which are now used as natural-gas mains. It is obvious that these mains may be used for piping retort-oven gas to the various industries for industrial purposes and to the cities for domestic use. It is also obvious, since the anthracite field has approximately reached its peak of production, that small coke made at by-product ovens will be used more and more to replace hard coal west of a line north and south running through Buffalo. It is the opinion of the author of this paper that the majority of the retort-oven plants will be placed in the coal field on account of the fact that the pipe lines are already established and on account of the further fact that by-products such as tar, ammonia, and benzol may be shipped from the Pittsburgh district with as much economy as from any other point. Freight rates throughout the country have been established so that the cost of transporting a ton of coke to market from the Pittsburgh district is about equivalent to transporting $1\frac{1}{2}$ tons of coal. This makes it possible to ship coke to the various points as cheaply as the coal necessary to make a ton of coke, so that there is no transportation advantage in shipping coal to a distant point to make by-product coke. The modern construction of railroad coke equipment has reached a stage that enables the railroads now to haul a train of coke as cheaply as a train of coal of the same weight, the difference, if any, being very slight. The tendency, therefore, will be to adjust coke freight rates on the basis of cost of transportation, which will be in favor of establishing coke plants in the coal field in the Pittsburgh district.

With this adjustment in view, it would be more profitable to establish the by-product-coke plants in the coal field, especially if the gas could be sold to the pipe lines now established. Or if the railroad lines are equipped for electric traction, largely, it may be that it would pay to put the by-product-coke plants in the coal field and by the installation of gas engines at the coke plants generate electric power which would be transmitted and sold to the railroads at low prices. The question of where to establish the by-product-coke plant will depend also largely on whether the owner of the coke plant uses his own products or whether the coke plant is established for the general market. It would seem, however, that a commercial coke plant would have a larger market both for gas and coke if established in the Pittsburgh coal field than if placed elsewhere, as the established freight rates on coke would allow shipments of coke to any point now available, and the local market for gas would absorb at fair prices the surplus gas of the plant.

The Book Cliffs Coal Field, Utah

BY ROBERT S. LEWIS, SALT LAKE CITY, UTAH

(Pittsburgh Meeting, October, 1914)

THE Utah coal field to which the name Book Cliffs is applied runs in a northeast direction from Mt. Hilgarde, in Sevier county, along the escarpment of the Wasatch Plateau to the vicinity of Castle Gate, thence southeast along the Book cliffs, which lie to the north of and generally within sight of the Denver & Rio Grande Railroad, to the Colorado line. The Book cliffs form the southern escarpment of the West and East Travaputs Plateaus, and vary in height from 2,000 to 6,000 ft. At the base of the Wasatch and Book Cliffs escarpments is a broad plain known as Castle valley. The coal beds outcropping in the escarpments along this valley are worked on the largest scale of any in the State, and this particular area is known locally as the Castle Valley coal field.

Geology[1]

The Book cliffs are capped by the Tertiary—Eocene—formation, consisting of varicolored shales, buff sandstone, conglomerate, and subordinate thin beds of limestone. Different sections show diverse stratigraphy. This is mainly the Wasatch formation, and lies unconformably upon the rocks of the Upper Cretaceous. Fig. 1 is a generalized section of the rocks of the Book Cliffs field. The Mesaverde formation contains the cliff-making rocks of the Book cliffs, and is composed of massive sandstone interbedded with sandy shales and beds of coal and carbonaceous shale. The sandstone has greater resistance to erosion and the formation is characterized by escarpment ledges and cliffs. The shale is commonly sandy and is drab in color, but where it is associated with the coal it is usually carbonaceous. The sandstone is generally buff, though occasionally it is white, and in places red. The bedding ranges from thin to massive. In the lower part of the formation the sandstone is red in

[1] Abstracted from *Bulletin No.* 371, *U. S. Geological Survey*, by G. B. Richardson. Late information indicates the following as the more correct geologic column for the rocks in this field: Wasatch remains as given. Mesaverde to be replaced by Laramie with average thickness of 1,800 ft. In place of Mancos shale, the upper 400 ft. is Montana shale and the next 2,600 ft. is Colorado shale. The Dakota and Jurassic remain as given.

many places, due to the burning of the underlying beds of coal. The Mancos shale is a fissile black, blue, gray, and drab clay shale, which contains local lenses of limestone and, at the top, thin beds of buff sandstone, by which this formation grades into the Mesaverde. The shale is much broken by cracks and joints, which frequently contain saline films. It forms the base of the Book cliffs and underlies Castle valley.

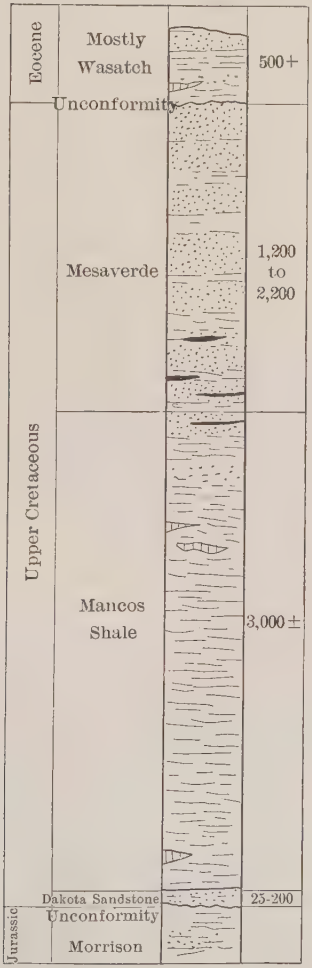


Fig. 1.—Generalized Section of Rocks in Book Cliffs Field.

Coal

The coal beds are found in the lower 700 ft. of the Mesaverde formation, and at the top of the Mancos shale. The coal is a high-grade bituminous coal, that can be stored for a long time without deterioration. Some of the coal which had been exposed to the weather for more than a

year appeared to be in excellent condition. Cleating is often strongly pronounced and is taken advantage of in mining. Patches of resin occur in the seams, being much more plentiful in some mines than in others. The coal burns freely, leaving a moderate quantity of ash of a character closely resembling wood ashes. British thermal units range from 13,480 to 14,750. The seams worked vary from 5 to over 20 ft. in thickness, and are remarkably free from impurities. They dip at an angle of 6° or less, and the mines are usually dry—so much so that the companies have found it necessary to install elaborate sprinkling systems to wet the dust, which is of a highly inflammable nature. All the mines in the district are worked on the room-and-pillar system. Entries are driven on the double-entry plan. Mining machines have been in use for only a few years, but have come into such favor that they will soon be in universal use throughout the field. At nearly all the mines the seams outcrop high up on the steep hillsides, making it necessary to use inclines for transporting the coal to the tipples. A striking feature of the field is the great extent of the area of burned coal, which, in some cases, reaches to a distance of 2,000 ft. from the outcrop.

During 1913, Utah produced 3,289,265 tons of coal. Imported coal was consumed to the amount of 326,095 tons, most of this coal coming from Wyoming. The coal exported from the State amounted to 680,091 tons, which was distributed among the States of Washington, Oregon, California, Nevada, Montana, and Idaho. Most of the 314,694 tons of coke made at the Sunnyside plant was shipped to the smelter at Anaconda.

Operators

The largest operations in this field are those controlled by the Denver & Rio Grande Railroad and the United States Smelting Co. The former controls the mines at Sunnyside and Castle Gate, as well as the Utah, Winter Quarters, Clear Creek, and Schofield mines in the Pleasant Valley coal field. The latter controls the mines at Mohrland, Black Hawk, Hiawatha, and the Panther mine near Castle Gate. The mines at Storrs, Standardville, and Kenilworth belong to other companies. All of these mines have railroad shipping facilities. The road from Price to Black Hawk, the Southern Utah & Castle Valley Railroad, has a 5 per cent. grade in places, which is too steep for good railroading and the economic handling of the large tonnage expected from this district. For this reason the United States Smelting Co. is building a new road, the Utah Railway, from Black Hawk, to tap the main line of the D. & R. G. R. R. at a point about a mile south of Castle Gate. This line will be 24 miles long, and will have a maximum grade of 2 per cent. Construction is well along, and the road will probably be in operation some time this fall. The

accompanying sketch map, Fig. 2, shows the important mines in both the Book Cliffs and Pleasant Valley fields.

Important Coal-Producing Properties of Utah.[a] *For Year* 1913

Name of Mine	Tons	Men Employed	Operator	County	Post Office
Winter Quarters...	400,214	368	Utah Fuel Co.....	Carbon...	Winter Quarters.
Clear Creek.....	272,363	225	Utah Fuel Co.....	Carbon...	Clear Creek.
Utah Mine.....	86,528	117	Utah Fuel Co.....	Carbon...	Utah Mine.
Castle Gate.....	319,692	502	Utah Fuel Co.....	Carbon...	Castle Gate.
Sunnyside.....	669,907[b]	955	Utah Fuel Co.....	Carbon...	Sunnyside.
Kenilworth.....	402,952	405	Independent C. & C. Co...	Carbon...	Kenilworth.
Aberdeen.....	470	2	C. A. Ballinger.....	Carbon...	Price.
Carbon.....	9,019	290	Panther Coal Co.....	Carbon...	Castle Gate.
Storrs.....	116,977	268	Spring Canyon Coal Co....	Carbon...	Storrs.
Hiawatha.....	395,185	461	Consolidated Fuel Co.....	Carbon...	Hiawatha.
Black Hawk.....	176,349	308	Black Hawk Coal Co.....	Carbon...	Black Hawk.
Ketchum.....	500	15	Ketchum Coal Co.....	Carbon...	Castle Gate.
Standard.....	10,000	56	Standard Coal Co.....	Carbon...	Helper.
Neslen.....	50,266	91	American Fuel Co.....	Grand....	Neslen.
Mohrland.....	262,359	357	Castle Valley Coal Co.....	Emery....	Mohrland.
Johnson.....	620	2	A. M. Johnson.....	Emery....	Orangeville.
Black Diamond...	262	2	Black Diamond Coal Co...	Emery....	Orangeville.
Rock Canyon.....	417	2	Akel Anderson.....	Emery....	Clawson.
Queatchappel.....	794	2	San Rafael Fuel Co.....	Emery....	Castle Dale.
New York.....	960	3	Utah Fuel Co.....	San Pete..	Fairview.
Larson.....	1,200	4	Mt. Pleasant Coal Co.....	San Pete..	Mt. Pleasant.
Morrison.....	2,100	6	.....	San Pete..	Sterling.
Wasatch.....	16,701	35	Weber Coal Co.....	Summit...	Coalville.
Grass Creek.....	56,163	80	Union Fuel Co.....	Summit...	Grass Creek.
Rees.....	22,879	29	Rees-Grass Creek C. C....	Summit...	Coalville.
Briquette.....	2,800	7	Superior Fuel & B. Co.....	Summit...	Coalville.
North Star.....	750	3	Geo. Gray.....	Uintah....	Vernal.
Hatch.....	1,100	4	P. E. Ritch.....	Uintah....	Vernal.
Mill.....	3,100	10	Pack & Allen.....	Uintah....	Vernal.
Kidd.....	397	3	A. C. Kidd.....	Uintah....	Vernal
	3,283,024				

[a] Compiled from the 1913 report of J. E. Pettit, State Coal Mine Inspector.
[b] Nearly all of this coal was utilized in making 314,694 tons of coke.

Castle Gate Coal Co.

Two mines of the Utah Fuel Co. are at Castle Gate. The No. 1 mine entrance is about 600 ft. southwest from the tipple, while the No. 2, or Willow Creek, mine is about a mile to the northeast, being reached by a motor haulage road 5,300 ft. long. This road has a gauge of 3 ft. 4 in., and is laid with 60-lb. rails. Trolley supports, made of $1\frac{1}{2}$-in. pipe, set in concrete, are spaced 25 ft. apart. The road, average grade 2 per cent., passes through two tunnels, one of which is 1,000 ft. long and the other 400 ft.

Tipple.—This is of steel construction, and was built by the Ottumwa Box Car Loader Co. It has a capacity of 2,000 tons per day, shaking

screens being used. The sizes of the various products are as follows: Dust, all coal passing a $\frac{3}{8}$-in. screen; screened slack, from $\frac{3}{8}$ to $1\frac{5}{8}$ in.; nut coal, from $1\frac{5}{8}$ to either 4 or 6 in.; lump coal, all over 4 to 6 in. For some years the slack coal, that passing a $1\frac{5}{8}$-in. screen, was sold for heating and power purposes until some one suggested taking out the dust and making two products. This movement, once started, had to be followed by all the companies, and the result is that the dust has practically no market and is usually sent to the dump.

As the two mines are on opposite sides of the canyon through which the railroad runs, the tipple is provided with two Phillips cross-over dumps, one for each mine. Ottumwa box-car loaders are employed for loading the nut and lump coal in box cars; the open cars are loaded directly from chutes.

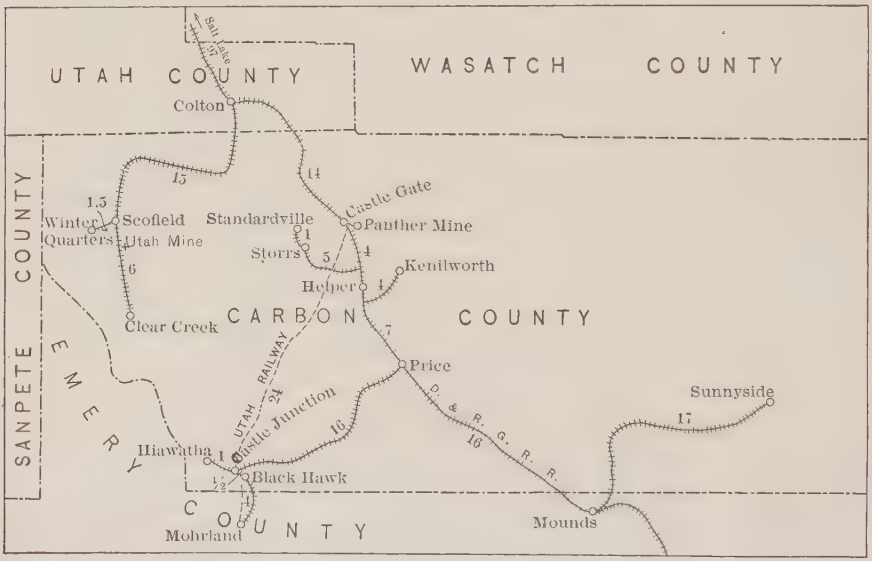


Fig. 2.—Map showing Location of Important Mines in the Book Cliffs and Pleasant Valley Coal Fields, Utah.

Power House.—Dust is used for fuel under the eleven 125-h.p. fire-tube boilers. A drag conveyor brings the dust from the tipple to a 100-ton steel bin just outside the power house. From the bin, the coal is conveyed to the bunkers above the boilers, which are fed by American automatic stokers. The generators installed furnished direct current at 500 volts. These are now being replaced by units consisting of a 500-kw. General Electric alternating-current generator supplying current at 2,300 to 4,000 volts and running at 3,600 rev. per minute, directly connected to a Curtis turbine taking steam at 100 lb. and running condensing. One unit is now in operation, and the foundation for a second

is about completed. Current will be transmitted to the mines at 4,000 volts, where motor-generator sets will transform it into direct current at 500 volts for the locomotives and mining machines.

The A or lowest seam, 4 to 10 ft. thick, was the first one opened, and this is now worked out, save for the stumps of a few pillars. The B seam, separated from A by a 30-ft. parting, is not worked in this mine; the seam is too thin (3 ft.) and the coal is not very clean. Most of the present output comes from the C seam, 4 to 7 ft. thick, 100 ft. above. A 4-ton electric locomotive is used in this seam to take the coal to the head of the incline, from where it is dropped to the main haulage slope running abong the A seam. The D seam, 7 to 11 ft. thick and underlain by 15 to 20 ft. of sandstone and shale, was discovered only a short time ago and the work done there is in the nature of development work. At present this seam is reached by an incline driven up from the C seam. This necessitates a rather roundabout method of getting the coal to the tipple, so the company is considering the driving of a slope from the D seam to the surface and dropping the coal to the tipple by a new incline.

The methods of mining differ in minor details in the two mines, but the following description of the working of the C seam of the No. 1 mine will give a good idea of the method employed. Raise entries are run directly up the dip, on a grade of 10 per cent. The first rooms are run parallel to the raise entries, leaving a barrier pillar about 70 ft. thick between. The side entries are turned off at right angles, giving a practically level grade. The rooms are turned off at an average angle of about 55°, which gives a grade of about 5 per cent. The rooms for hand mining are 20 ft. wide, while those for machines are driven 30 ft. wide. The pillars between the rooms are 40 ft. wide. Barrier pillars 75 ft. thick are left between the ends of the rooms and the air course of the next entry. Though all workings are surveyed, the miners are often instructed to start the last break-throughs between the rooms, by driving at right angles to the rooms at a point 40 ft. back from the end. The State law requires connections between the rooms every 125 and 159 ft. and every 100 to 125 ft. between entries. A large part of the mining in this mine is done by hand, but recently five Sullivan short-wall machines have been introduced. Little or no timber is required in the rooms, but the roof must be carefully supported when the pillars are being drawn. A thorough system of electric shot firing has been installed. Double switches, to guard against leakage, are used wherever the firing wires cross the trolley line. Inspectors examine all places where shots have been fired in order to locate any missed shots. Mules are used for haulage in the rooms.

The No. 2 mine has two workable seams; the upper one corresponds to the D seam in No. 1 mine, and is from 18 to 25 ft. in thickness; the other corresponds to the B seam and is from 4 to 7 ft. in thickness. All mining

is done by machines, two Jeffrey and three Sullivan machines of the short-wall type being used.

A General Electric locomotive, originally of 20 tons weight but reduced to 15 tons, takes the trips to the tipple. Mules are used underground for gathering and haulage in the side entries.

The output of No. 1 mine is about 850 tons per day, and of No. 2 mine from 600 to 700 tons per day. Wooden cars with end dump are used. These have a 28-in. wheel base, 18-in. wheels, and are made of rough 2-in. plank. Two sizes are employed; one holding 2,500 lb. and the other holding 5,800 lb.; the average monthly capacity of all cars is about 2,300 lb. The smaller cars are used in the narrow seams where head room is limited.

The Spring Canyon Coal Co.

The mine office of this company is at Storrs, Carbon county, Utah, which is reached by a branch railroad 4½ miles long, connecting with the main line of the Rio Grande Western at a junction point about a mile north of Helper.

To one familiar with the hastily constructed buildings found in so many Western mining camps the town of Storrs is a pleasant surprise. Laid out along the bottom of the canyon, the office, hotel, and miners' cottages are supplied with bath, toilet, and electric lights, and are built of light-colored sandstone taken from the nearby quarry of the company.

The tipple and power house, shown in Fig. 3, are at the lower end of the town. The tipple is of steel construction and is equipped with shaking screens. The coal passes from the receiving hopper to the screens over a steel apron picking conveyor 5 ft. wide. Men on each side of the conveyor pick out the waste or "bone" as the coal passes by. When the mine is working at full capacity there is a depth of from 15 to 20 in. of coal on the conveyor, making it hard to clean the coal thoroughly. To overcome the difficulty, the management has planned to move the conveyor to the lower end of the screens, where only the nut and lump coal will pass over it. The width will be increased to 7 ft. and a central longitudinal division will be introduced to keep the two sizes of coal separate. Although the coal is no dirtier than other coals in this field, this company is the first to install a picking conveyor. The screens are arranged to give the following products: Slack, all the coal passing a 1⅝-in. opening; nut coal, between 1⅝ and either 4½ or 6 in.; lump coal, all over 4½ or 6 in. Some contracts specify 4½ in. as a maximum size for nut coal, while others call for coal as large as 6 in.; so two sets of screens are required. The slack is elevated to the top of the tipple, where it passes through a revolving screen with ⅜-in. openings. The oversize, or "screened slack," is sold, but there is so little demand for the undersize, or "dust," that it usually goes to the dump. The run-of-mine coal gives

about 45 per cent. lump, 30 per cent. nut, and 25 per cent. slack; the dust in the slack being from 5 to 10 per cent. of the total coal mined. The tracks under the tipple are laid on a 2 per cent. grade so that the cars will move under the action of gravity when the brakes are released. Two tracks are for lump coal and one each for the nut coal, screened slack, and dust. Gondola cars are loaded directly from chutes, but the box cars are loaded with lump or nut coal by means of Ottumwa box-car loaders.

A narrow slot is cut in the steel chute under the $1\frac{5}{8}$-in. screen. This permits of sufficient slack passing through to supply the power house with fuel. A spiral conveyor carries the slack to the boiler room, where it is fed by hand to five 150-h.p. Springfield boilers. These boilers are of the marine type and give excellent satisfaction. Two 200-kw. General

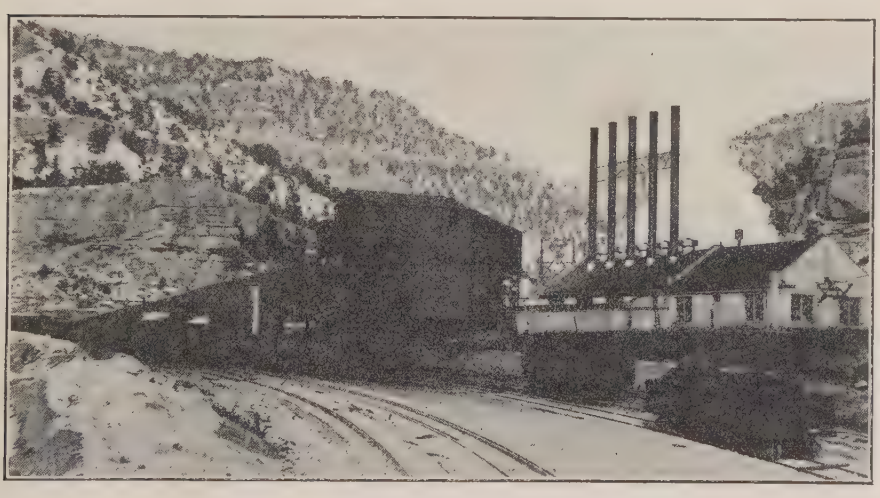


FIG. 3.—POWER HOUSE AND TIPPLE OF THE SPRING CANYON COAL CO. SHOWS CHARACTERISTIC TOPOGRAPHY OF THE BOOK CLIFFS COAL FIELD.

Electric alternating-current generators, running at 200 rev. per minute, furnish current at 2,300 volts. They are directly connected to Buckeye engines. For supplying lights and extra power, when needed, there is a 40-kw. General Electric generator running at 300 rev. per minute and delivering current at 2,300 volts. This is directly connected to a B. F. Sturtevant vertical engine. The power is transmitted to the mine, where a motor-generator set transforms it into direct current at 250 volts.

The workable coal on the property is found in three seams. The lowest, or No. 1, seam is 8 ft. thick, and the more extensive workings are in this seam. Above this seam, and separated from it by a 20-ft. shale parting, is the No. 2 seam, $4\frac{1}{2}$ ft. thick. This seam is not being worked at present. The No. 3 seam, which varies in thickness from 6 to 12 ft., is 180 ft. above the No. 2 seam. The outcrop of the coal is high up on the

steep mountain side, the seams dipping back into the mountain at an angle of from 7° to 10° from the horizontal; direction of dip is about N. 18° E. The seams are exceptionally free from slate. Near the mouth of the main entry the No. 1 seam pinches down to an unworkable thinness, but, fortunately, this irregularity is of small extent. The main haulage way runs NW-SE. to reduce the grade to a maximum of 3 per cent. The locomotive pulls the loaded trip up to the head of the slope. From there it pushes the cars out to the loading station of the aerial tramway, which connects the mine with the tipple. The loading station is close to the entrance to the mine. The necessity of pulling the loads up grade introduces an element of danger, for a loaded car or the entire trip might break away and, running down the slope, cause a serious accident. The company has installed a derailing switch, which is operated by a rope running along the side of the haulage way. Should the cars break away, the trip rider can pull the rope opening the switch and thus derail the cars. Two trips have been stopped in this way, and what promised to be serious accidents were prevented. The No. 3 seam is reached by a haulage road 2,500 ft. long, which runs along the side of the mountain. This seam has only recently been opened. At present it is producing 400 tons a day, while the No. 1 seam is sending out 700 tons per day, making a total of 1,100 tons as the daily output of the property.

The side entries are run nearly east-and-west to give a grade of from $2\frac{1}{2}$ to 3 per cent. Mining is done on the room-and-pillar system. The rooms are 300 ft. long by 30 ft. wide, with 34-ft. pillars between. A barrier pillar 50 ft. wide is left between the ends of the rooms and the air course of the next entry. The first rooms were laid off directly on the pitch of the coal, giving a grade of from 7 to 10 per cent. This was done to take advantage of breaking the coal on the face cleats, but the grade was found to be too steep for safe and easy handling of the cars. The rooms are now laid off at an angle with the entry, so that the grade of the track along the rooms is about 3 per cent. Horses take the cars to the side entries. Gathering locomotives of the reel type then pull the cars to the main haulage way. The entries are 15 ft. wide and 8 ft. high. The coal is undercut by machines, and is broken down with No. 3 Guardian powder, made by the Independent Powder Co.

Two General Electric locomotives, of the outside-frame type and weighing 15 tons, are used on the main haulage way, which has 50-lb. rails laid on heavy sawed ties. Three gathering locomotives are used. They weigh 6 tons each. All the locomotives have a 5-ft. wheel base, and operate on 250 volts. Large horses are used in the rooms, because of the ample head room afforded and the great weight of the cars.

The following mining machines are used, all of the short-wall type: One Jeffrey, four Sullivan, and seven Goodman. Great care is exercised to keep the faces of the rooms as square as possible. In making the sump-

ing cut the machines are pointed outward to cut out into the rib, so that the full width of the room is easily broken. The rib is left slightly irregular, but is squared up with little trouble. The machines make a 6-ft. cut. The blasting holes are put in to a depth of $5\frac{1}{2}$ ft., the coal breaking readily to the full depth of the cut.

Sturtevant fans, 5 ft. by 2 ft. 6 in., are used for ventilation, one for each seam. Owing to the small extent of the working, no gauge pressure can be noticed. The overcasts are made with walls 10 in. thick, the bottom being reinforced with old rails. A mixture of 1 part cement with 5 parts of "bug dust," or machine cuttings, makes an excellent material for the walls. It is also used for the stoppings.

The tramway, connecting the mine and tipple, was built by the American Steel & Wire Co. It has a drop of approximately 325 ft. in a distance of 3,000 ft. The loading station is on a precipitous slope of the mountain. This location is necessary in order to avoid an angle station on the tramway. A steel truck holding two of the tramway buckets is used in place of a regular mine car. The weight of the assembled empty car is 3,300 lb. and each bucket holds on an average 2,300 lb. of coal, making the weight of a loaded car 7,900 lb., of which 41.8 per cent. is dead weight. This unduly large per cent. of dead weight makes the cost of haulage so high that the company is now designing a lighter car. The loaded cars, as they come from the mine, pass over a Strate coal scale and then on to a tipping platform, where the tramway carriers, or hangers, are attached to the lugs on the buckets. The steel truck is then lowered to free the buckets and to permit of their being started on their way. Two empty buckets are now run over the truck, which is raised, and the hangers are then disengaged. The assembled empty car is kicked on to the return track by the oncoming load. A 6-h.p. motor operates the tipping platform. The tramway is self operating, but a 30-h.p. motor is installed at the head end to bring up supplies in case no coal is being sent down. At the tipple, the buckets are dumped by hand. The carriers are automatically detached from the cable and run into the tipple on a track. On making the half-circle turn to the return track they pass over a small bin, or hopper. The dumping has to be done quickly, because of the limited dumping space, and the buckets must not come too close together. The estimated capacity of the tramway is 250 tons per hour, but this is not attained at present. It is planned to extend the dumping space so that the full capacity can be had.

The dust in this mine is highly inflammable. To minimize the danger of explosion an extensive sprinkling system has been installed, by means of which all haulage ways and working places are kept damp. The workings are free from gas and the miners use acetylene lamps. The mine is equipped with the Stromberg-Carlson telephone, and the haulage ways are lighted by electricity. Near the entrance to the mine is a repair

shop, which is provided with emery wheel, forges, drill press, lathe, and shaper.

Analyses of Coals

Seam	Volatile Matter Per Cent.	Ash Per Cent.	Moisture Per Cent.	Fixed Carbon Per Cent.	Sulphur Per Cent.	B.t.u.
No. 1	46.7	4.5	1.9	46.9	1.07	14,547
No. 2	47.4	4.6	0.7	47.2	0.98	14,730
No. 3	43.4	3.6	1.8	51.2	0.89	14,725

The mine has been producing but slightly more than a year. The company controls 2,000 acres, of which 1,200 acres are coal bearing. The elevation of the mine is 6,780 ft.

Standard Coal Co.

This mine is at Standardville, about a mile northwest of Storrs. The Standard is one of the newest companies in the field, and has not yet

FIG. 4.—REINFORCED CONCRETE RETAINING WALL AND SHAKING-SCREEN FOUNDATION AT THE MINE OF THE STANDARD COAL CO.

reached the producing stage. The work being carried on is development work, though daily shipments of from 350 to 400 tons are being made.

The mine and tipple are connected by an incline 4,200 ft. long, the first half of which is on a 7 per cent. grade, while the second half has a 12.5 per cent. grade. At the entrance to the mine the coal seam is 14 ft. thick, but this represents a rather unusual condition, as the average thickness is from 10 to 12 ft. A thin parting occurs about 5 ft. from the floor.

The reinforced concrete tipple, Fig. 4, presents some interesting features of design. In order to dispense with the approach trestle, and to give

a substantial bracing to the tipple, a reinforced concrete retaining wall, Fig. 5, was built. The wall is 18 in. thick at the bottom, 10 in. thick at the top, and is 46 ft. high at the highest point. Buttresses are placed every 13 ft. The mine cars pass continuously in one direction over a track laid close to the edge of the wall. The filling required was 12,000 cu. yd.,

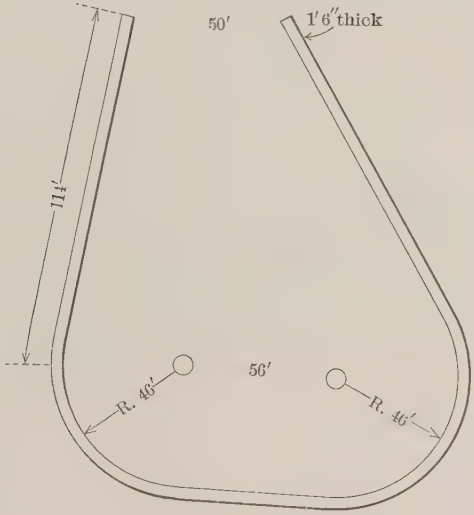


FIG. 5.—PLAN OF REINFORCED CONCRETE RETAINING WALL SHOWN IN FIG. 4.

much of it coming from the cuts made during the construction of the incline.

The shaking screens rest on roller bearings, which are secured to heavy I-beams imbedded in the inclined concrete stringers. Pillars, placed be-

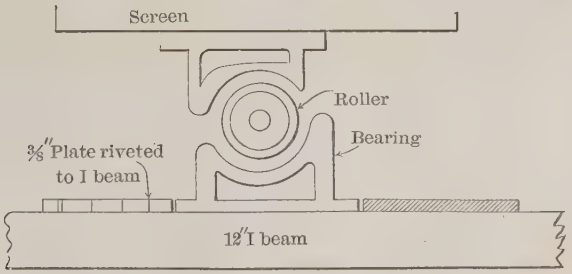


FIG. 6.—BEARING FOR SHAKING SCREENS.

tween the railroad tracks, support the stringers and screens. To secure the undulating motion of the suspended type of shaking screen, special bearings were designed. Their arrangement is shown in Fig. 6. The brace plates, placed against the sides of the bearings, are firmly riveted to the I-beams, which have their ends strongly anchored in the retaining

wall in order to make the structure as rigid as possible. It is said that the vibration, when the screens are running, can scarcely be noticed.

The slack is elevated to a revolving screen, which discharges the dust and screened slack into steel bins placed beneath the screen. A rotary dump, made by the Link Belt Co., is used for the mine cars. The railroad box cars are filled by means of an Ottumwa rocking cradle loader, the first one of this type to be used in the State.

Independent Coal & Coke Co.

This company's mine is at Kenilworth, the terminus of the Kenilworth & Helper Railroad, a branch line 4 miles long, which leaves the main line of the D. & R. G. R. R. at Kenilworth Junction, a point about a mile south of Helper. The road has a maximum grade of 6.4 per cent., requiring the use of Shay locomotives, three of which are now in operation. Two of these are of 90 tons weight and the other weighs 70 tons.

Kenilworth is a well-built town, laid out in a systematic manner on a gently sloping flat at the foot of the mountains. There are 159 buildings in all, 135 of which are for the miners. Most of these latter are four-room houses. About 400 men are employed at present.

Power House.—This is a substantial stone building containing six boilers, feed-water heaters, pumps, and generator. The boilers are 125-h.p. Kewanee horizontal tubular boilers, 72 in. by 16 ft., and are equipped with American automatic stokers. Dust is used for fuel. The 500-kw. alternating-current generator is of General Electric make, operating at 2,300 volts, and is directly connected to a Curtis turbine running at 3,600 rev. per minute and taking steam at 125 lb. The exhaust steam goes to a No. 2 Worthington counter-current jet condenser and then to the cooling apparatus, consisting of 24 Schute Koerting spray nozzles and a concrete reservoir of 100,000 gal. capacity.

Tipple.—The old wooden tipple is to be superseded by an all-steel tipple of 3,000 tons daily capacity, designed by the Link Belt Co. This tipple is of the shaking-screen type and will make five sizes of coal, in addition to the straight mine run. Two 60-ton storage bins are provided for the screened slack and dust. A drag conveyor will take the dust to the power-house storage bins for use under the boilers. Ottumwa box-car loaders will be used. One, of the standard pusher type, is equipped with horizontal steam engines; the other is an Ottumwa-Ecks loader of the belt-conveyor type, illustrated in Fig. 7. This is equipped with an electric motor and is so arranged on a turntable that it can be used on either of two tracks. The dump is of the Link Belt revolving type and handles one mine car at a time. One feature of the tipple is the use of an incline car lift, shown in Fig. 8. The empty cars, upon passing the cross-over switch and starting back, run on to a short stretch of steeply inclined track, up

which they are pushed by an endless chain, running in the middle of the track, which has attached lugs that catch against the car axles. By the

FIG. 7.—OTTUMWA-ECKS ELECTRIC BELT-CONVEYOR BOX-CAR LOADER AT THE MINE OF THE INDEPENDENT COAL & COKE CO.

use of this lift the head car of the empty trip is brought to the same level as the rear car of the loaded trip, and the trip rider can change the haulage

FIG. 8.—LINK BELT ROTARY-DUMP TIPPLE.
Car lift shown in lower left-hand corner.

cable from the loaded to the empty trip without being compelled to carry it from one level to another.

Mine.—Two inclines are used to drop the coal down from the mine to the tipple. The first, or Kenilworth, incline, beginning at the tipple, has a length of 200 ft. on a 2 per cent. grade, then a stretch of 620 ft. on a 27.7 per cent. grade, the last part being 1,290 ft. long with a grade of 39 per cent. This incline delivers the empty cars to the second incline, called the "shelf road." This incline is 1,200 ft. long, with an average grade of 12½ per cent. It runs around the edge of the mountain to the mouth of the mine, following the dip of the sandstone strata. A 25-h.p. motor is used on the main incline for starting the trips, while the shelf road is provided with a 200-h.p. motor. Eight cars make up a trip. The cars are of wood and weigh 3,500 lb. each. Their average load is 7,000 lb. of coal, the cars being piled high with lumps. There are three workable seams on the property. The lowest seam averages 18 ft. in thickness, and is called the Aberdeen seam. The Royal Blue seam, 7 ft. thick, is 45 ft. above this. The Kenilworth seam is 66 ft. above this and has an average thickness of 18 ft. The present daily output of 1,500 tons comes almost entirely from the Aberdeen seam. This seam dips N. 10° W. on a 10 per cent. slope. The main haulage way runs N. 32° 58′ W. along the seam on a 9 per cent. slope for 2,200 ft. and then turns due north to keep within the property lines. This slope will soon be double tracked throughout. It is provided with a double-drum 400-h.p. electric hoist, operating at a speed of 1,000 ft. per minute. Rooms are 350 ft. long and 24 ft. wide, with 50-ft. pillars between. Entries are driven 7 ft. by 12 ft., each pair being driven 65 ft. center to center. Horses are used for gathering, no locomotives being employed at present, though their use in the future is being considered. The coal is mined both by hand and by machines, six of which are in operation, one Jeffrey and five Sullivan, all of the short-wall type. Monobel powder is employed for breaking the coal, and all shots are fired by electricity. An elaborate sprinkling system has been installed. No timber is required, the mine roof being exceptionally solid. No work has been done toward the opening of the Royal Blue seam, and development on the Kenilworth seam has been in progress for only a few months. To reach these seams a connecting slope has been driven from a point at the bend in the main haulage slope of the Aberdeen seam. This connection is 540 ft. long and has a grade of 15 per cent. All the coal from these upper seams will be dropped to the bottom seam for hauling to the surface. The double tracking of the main haulage way will provide for a large increase in tonnage. A motor-generator set transforms the alternating current at 2,200 volts into direct current at 250 volts for mine use. The steepness of the main incline has been the cause of occasional accidents. The mine cars have broken away from the cable, but they hardly ever reach the tipple below, being stopped on a fill of soft earth about 200 ft. from the tipple. A large mass of sandstone broke away from the cliff above the incline and struck squarely on the rails of one track. It slid

down the track and smashed over the end of the old wooden tipple, doing considerable damage. To obviate any such damage to the new tipple, it is placed to one side of the old tipple and the tracks are curved for a short distance at the bottom of the incline. It is thought that the velocity of any runaway object will be so great that it will not make the turn, but will keep straight on past the tipple.

Analyses of Coals

	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Moisture, Per Cent.	B.t.u.
Aberdeen	43.97	51.62	2.96	1.45	13,480
Royal Blue	39.65	56.89	1.21	2.25	14,026
Kenilworth	42.83	52.52	2.28	2.37	13,536

Castle Valley Coal Co.

This company has its mine at Mohrland, in the northern part of Emery county, the southernmost point reached by the Southern Utah & Castle Valley Railroad.

Tipple.—This is a substantial framed structure, Fig. 9, covered with corrugated iron, and is built in the shape of the letter L; the long arm

FIG. 9.—TIPPLE AT THE MINE OF THE CASTLE VALLEY COAL CO.

contains the Phillips cross-over dump at the lower end; the upper part is long enough to hold a trip of 17 cars and serves to afford protection to the men in severe weather. The short arm contains the shaking screens. Wooden bins are used for the dust and screened slack. By their use, the tipple is not forced to shut down, should there be a shortage of cars, or difficulty in getting the loaded ones out of the way. The mine cars are

of steel and were made by the Watt Car Co. They weigh 3,300 lb. and hold from 3 to 4 tons of coal, depending upon the fullness of loading. They are equipped with link and pin coupling, but additional side chains are used for the sake of safety. Fairbanks quick-reading scales take the weights of the cars just before they run on to the dump. Christie box-car loaders, fitted with 50-h.p. motors, are used for loading the nut and lump coal. These work at a high rate of speed, throwing the coal against the ends of the cars with considerable force, which would lead to the making of a large amount of fines if the coal were very soft or brittle. The tipple has been in operation but a little over a year.

Mine.—The mine and tipple are connected by an incline 6,600 ft. long, having a maximum grade of 10 per cent. The tracks pass through two tunnels, one of which is 400 ft. long and is on a 4° curve. Sixty-pound rails are used, permitting of high speed of operation. The hoist, at the head of the incline, was built by the Denver Engineering Works. It was designed for lowering trips with a maximum rope pull of 15,000 lb. at a rope speed of 1,500 ft. per minute. This rope pull is taken care of by four band brakes, each 15 in. wide and 6 ft. in diameter. The brake rings are equipped with a large number of cooling vanes in order to radiate the heat generated. The rope sheaves are 6 ft. in diameter and are grooved for three laps of $1\frac{1}{8}$-in. steel rope. Owing to unfavorable grades, the hoist is not self-starting, but is geared to a 170-h.p. motor. This is thrown out of gear as soon as the trip is well under way. The length of the hoist is 20 ft., width 10 ft., and height 7 ft. The shipping weight, exclusive of motor, was 55,000 lb. The hoists used at the Black Hawk, Hiawatha, and Standard mines are of similar design. The coal seam on this property varies from 12 to 19 ft. in thickness, and dips S. 10° 30′ E. at a slope of 2.85 per cent. The main haulage way runs N. 72° 02′ W. and has a slope of 0.3 per cent. in favor of the loads. Side entries are run N. 18° 58′ E. and 18° 58′ W. on grades of +1.95 per cent. and −2.45 per cent. respectively. Rooms are 400 ft. long by 20 ft. wide, with 40-ft. pillars between. Entries are driven 8 by 10 ft., with 50-ft. pillars between. Where the seam is thickest, the first cut is mined 6 ft. in height; a second cut of from 4 to 6 ft. is then taken out, and the remainder of the seam, left to insure a good roof, is taken down when the pillars are pulled. It is the intention of the management not to pull any pillars until all the rooms have been worked out. The pulling of the pillars will then begin at the boundary of the property.

Mining is done wholly by machines. At the present time five Goodman chain breast machines and four Jeffrey short-wall machines are used. Hauling is done by Goodman locomotives, of weights varying from 5 to 10 tons. These are equipped with reels for gathering purposes. Because of the highly inflammable nature of the dust, the mine is equipped with a well laid out sprinkling system. The main entries have 3-in. pipes, with

2-in. branch lines down the side entries. A $\frac{3}{4}$-in. pipe runs into the rooms and 500 ft. of hose are connected to the pipe. A complete electric firing system has been installed. The powder used is 40 per cent. Hercules dynamite. Ventilation is good, but the company has recently ordered a Jeffrey fan of 300,000 cu. ft. per minute capacity against 3 in. of water. Power is generated at 250 volts, direct current, and is transmitted as such to the mine.

The coal is quite free from impurities and breaks readily and in large lumps, the output running nearly 70 per cent. lump. The daily production is from 1,600 to 1,700 tons. The elevation of the mine is 7,800 ft.

Black Hawk Coal Co.

The camp of Black Hawk is about 4 miles north of Mohrland. The main yards of the new Utah Railroad will be located here and an elaborate system of tracks is being laid out.

Tipple.—This is a frame structure built a few years ago. An incline laid with 45-lb. rails and 7,400 ft. long, with a drop of 1,000 ft., connects the mine and tipple, yet the coal is discharged near the bottom of the tipple on to an apron conveyor which elevates it to the screens. This conveyor is a source of much trouble; so much so that the present company, which inherited the tipple, is drawing plans for a new steel tipple. An interesting feature of the present tipple is the use of lip screens with openings 3 in. wide at one end, 5 in. wide at the other, and 11 in. long. This type of screen is much used in the Wyoming coal fields. Its particular advantage lies in the fact that the pieces of coal are rolled over in passing the screen, thus giving a more thorough screening and a better opportunity for the nut coal to pass through the screen. The sizes made at the tipple are the same as those made by the companies mentioned above. The company supplies much coal to the railroads. This coal is run-of-mine grade, but with a maximum size of 8 in. for lumps, as the firemen's union specifies that its members should not be compelled to shovel coal of a larger size. Though designed for handling 700 tons per day, the present daily output is from 1,100 to 1,200 tons. Box cars are loaded by steam-driven Christie box-car loaders. A Phillips cross-over dump and Fairbanks quick-weighing scales are used.

The coal seam varies in thickness from 15 to 22 ft. Mine cars are the Watt all-steel car. Haulage way and entries are driven 10 by 10 ft., with 40-ft. pillars between. The main haulage way runs S. 42° 37′ W and has a grade of −1.98 per cent. Side entries run as follows: First west entry, N. 79° 37′ W. on a −0.74 per cent. grade, then N. 47° 93′ W. on a +0.16 per cent. grade; first east entry runs S. 79° 37′ E. on a +1.29 per cent. grade, then N. 61° 57′ E. on a +2.27 per cent. grade. Rooms are 400 ft. long by 25 ft. wide, with 35-ft. pillars between. The

first rooms driven were turned off from the entries on too sharp a curve for electric haulage, but at the present time the rooms are turned off on easy curves and gathering locomotives will soon be used. Mules are used in the rooms, and 10-ton Goodman locomotives on the main haulage way. Mining is done by machines, eight Sullivan short-wall and eight Goodman breast machines being used. An electric firing system is installed. The switch at the mouth of each room is operated by the miners. Each entry has a switch at its head. This switch is in a locked box, which can be opened only by a shot firer. The main, or firing, switch is located outside the mine. All who enter the mine must hang up checks on a check board at the mouth of the mine, taking them up on their return. The shot firer is not allowed to close the main switch until every one has checked out. Should a check remain on the board, the man must be accounted for before the shots can be fired. First-aid sets are placed in the mine, usually near the heads of the side entries. Power is transmitted from the power house to the mine at 2,300 volts, where a motor-generator set transforms it into direct current at 250 volts. The elevation of the mine is 8,100 ft.

Consolidated Fuel Co.

The office of this company is at Hiawatha, about a mile northwest of Black Hawk. Two mines are operated, Hiawatha No. 1 and Hiawatha No. 2. They are on the same seam, but are on opposite sides of the canyon. The mines are reached by an incline 10,400 ft. long, 42-in. gauge, and with a maximum grade of 10 per cent. The seam is from 12 to 16 ft. thick. Rooms are 400 by 25 ft., with 45-ft. pillars between. Entries are driven 10 by 10 ft. The cabin pillars left at the ends of the rooms are 60 ft. wide, but are now being changed to 40 ft. The seam dips at about 3 per cent. and the entries are run on grades of from 1/2 to 2 per cent. The workings are carefully laid out from surveys, and this mine is operated in the most systematic manner of any in the district. Like the other mines of the district, it is provided with an electric firing system; 40 per cent. Hercules dynamite is used. Mining is done by machines; seven Sullivan and one Jeffrey short-wall machines and one Goodman breast machine are used. Hauling is done by four 10-ton and one 5-ton Goodman locomotives. The tipple has the usual arrangement of shaking screens. The dump is of the rotary type, being the first of this type used in the State. The daily output is from 1,500 to 1,600 tons.

The Castle Valley and the Black Hawk coal companies and the Consolidated Fuel Co. are controlled by the U. S. Smelting Co., which has plans for extensive improvements of the properties. The individual power plants are to be replaced by a central station located at Black

Hawk. Two 750-kw. generators, direct connected to steam turbines, are to be installed and sufficient space is to be left for two more. The tipple at Black Hawk is to be replaced by a steel one of 3,000 tons daily capacity. Its location is to be such that the tram line will be shortened to 6,000 ft. and will be in a straight line, instead of having a curve near the middle point as at present. The Hiawatha tipple is to be replaced by a steel one of 5,000 tons daily capacity. During the month of November, 1913, the payroll of these companies amounted to $107,000, about 1,800 men being employed.

Analyses of Coals

	Moisture, Per Cent.	Fixed Carbon, Per Cent.	Volatile Matter, Per Cent.	Ash, Per Cent.
Castle Valley	2.16	47.56	44.60	5.64
Black Hawk	2.14	47.29	43.53	7.04
Hiawatha	3.29	51.22	40.98	4.51

Utah Fuel Co.

The Utah Fuel Co. has two mines at Sunnyside. The seams are cut by a canyon, so two mines have been developed. Two seams are worked; the upper seam is 6 ft. thick and the lower seam is from 10 to 12 ft. in thickness. The parting is from 18 to 40 ft. thick in the No. 1 mine and from 12 to 30 ft. thick in the No. 2 mine. The coal dips a little north of east at an average of 9 per cent.

The main slopes are driven directly down the dip, and 200-h.p. hoists are used at the head of the slope. These are capable of handling trips of 16 cars, each car weighing 5,800 lb., at a speed of 1,500 ft. per minute. Side entries are turned off at right angles and at intervals of 400 ft., making the rooms 350 ft. long except where a fault of 30 ft. displacement cuts through the property; the rooms driven to this fault are 600 ft. long. The rooms are turned off at 45° with the entries to reduce the grade. Horses and mules are used in the rooms and side entries, while electric locomotives haul the coal from the main slopes to the tipple. The two locomotives used at No. 1 mine weigh 8 and 13 tons, respectively, while the two at the No. 2 mine, where the length of haul is 2.65 miles, are of 13 and 15 tons weight. The Jeffrey fan at No. 2 mine is capable of supplying 300,000 cu. ft. of air per minute against 5 in. of water. Its motor takes alternating current at 4,000 volts directly from the transmission line, but a motor-generator set is used to supply direct current at 500 volts for the locomotives and mining machines. All mining is done by hand at present, but four Sullivan short-wall machines have been ordered.

This is a coking coal and practically the entire daily output of the mines, about 2,500 tons, is made into coke. The breaker, which also

contains shaking screens in case coal is to be shipped, is in two units, each of which consists of one set of toothed rolls, 36 in. diameter by 40 in. face, which break the coal to nut size (largest pieces 3 in.). From the rolls the coal goes to a No. 8 Williams Jumbo disintegrator. A Jeffrey continuous bucket elevator 36 by 12 by 12 in. elevates the coal to a revolving trommel, 5 ft. in diameter by 12 ft. long, with $\frac{3}{8}$-in. openings. The undersize goes to the oven bins, while the oversize is returned to the disintegrator. Six-ton electric larry cars are used for charging the ovens. There are 726 ovens in operation, but the company can hardly meet the demand for coke, most of which goes to the smelter at Anaconda. The ovens are of the beehive type, 12 ft. in diameter, and take a charge of 6 tons. The time of coking is 48 hr. and the coke made is 55 per cent. of the charge. All the coke is hand pulled.

The power house has 14 fire-tube boilers, of 125 h.p., 12 of which are kept in continual operation. Both alternating and direct current generators are used, but the company is planning to put all power on an alternating-current basis, and a 500-kw. turbo generator, similar to the one at Castle Gate, has just been installed.

Analyses of Coals

	Moisture,	Volatile Matter,	Fixed Carbon,	Ash,	Sulphur,	B.t.u.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
Castle Gate.....	2.88	39.57	54.41	3.14	0.493	13,698
Sunnyside.....	1.87	39.22	54.78	4.13	1.022	13,944

Acknowledgments

The photograph of the Standard Coal Co.'s tipple was taken by D. H. Blossom, chief engineer. I also wish to acknowledge my indebtedness to J. E. Pettit, State Coal Mine Inspector, Prof. R. B. Ketchum, of the University of Utah, and to the various officials of the coal companies through whose kind assistance I have been enabled to secure the data here presented.

An Aerial Tramway for Mining Cliff Coal

BY ARTHUR E. GIBSON, STORRS, UTAH

(Pittsburgh Meeting, October, 1914)

Synopsis.—A new feature in coal mining, where the coal is to be conveyed from a high to a lower elevation and the topography of the country is such as to preclude surface haulage.

The distance from the mine to the tipple in this instance is about 3,000 ft. and the difference in elevation between the terminals 321 ft., of which 225 ft. occurs in the first 1,300 ft. from the loading terminal.

A mine car was designed especially for this plant consisting of two tramway buckets on one pair of trucks, the buckets being lifted from the truck, attached to the tram line automatically, and conveyed to the tipple.

THROUGHOUT eastern Utah, in the Book Cliff range and the Wasatch plateau, the coal-bearing measures are admirably exposed, in the bold and for the most part bare escarpments. The geology is simple, the rocks being almost flat, or at the most dipping not more than 10 per cent.

The coal beds occur in a formation, or group of strata, consisting of sandstone and shale, 600 to 800 ft. thick, which has been correlated with the Mesaverde formation of the Cretaceous. Above this occurs about 2,000 ft. of alternate layers of buff-colored sandstone and shale.

Below the coal lie the Colorado shales, which are estimated to be about 1,500 ft. thick, below which are the red rocks of the Jura-Trias.

As a rule the coal beds all dip in a northerly direction and under the high mountain. The canyons opening out into what is known as Castle valley all cut the coal beds, and the beds rising in the direction of the valley are found frequently high up on the escarpment, often as much as 1,000 ft. above the valley.

The surface features of the region have an economic bearing on the availability of the coal. The valleys and gulches in the Colorado shale extend out from the coal areas across the edge of the plain, so that it is impracticable to build transportation lines parallel with the boundaries of the field within several miles of the coal land. It will be necessary, therefore, to build branch lines up to the coal lands from points on the Denver & Rio Grande Railroad traversing Castle valley.

Coal mines have been in operation in this region for the past 40 years,

some of the more important mines being: Winter Quarters, Castle Gate, Clear Creek, Sunnyside, Kenilworth, Hiawatha, Mohrland, Blackhawk, Storrs, Panther, Standard, Cameron and a number of smaller mines.

The total production of bituminous coal for the State of Utah during the year 1912 was 3,088,356 tons, practically all of which was from Carbon and Emery counties. The older mines were opened in the most favorable locations, *i.e.*, where the coal was low in the canyon, or where the bottom of the canyon intersected the coal floor and where an easy grade could be procured in building a railroad to the mine.

Of late, however, all of the mines have been opened at a high elevation

FIG. 1.—VIEW OF PART OF STORRS, SHOWING BOARDING HOUSE, STORE, AND ROCK COTTAGES.

above the level of the surrounding country, necessitating a system of haulage which would successfully lower the coal to a level which could be reached by railroad.

In most instances gravity planes have been installed, but at Storrs an aerial tramway is in operation.

This mine was opened in the fall of 1912 and is located in Spring Canyon, which opens upon Price river, 4½ miles west of Helper, where a branch leaves the main line of the Denver & Rio Grande Railroad. The branch railroad from Helper to Storrs, having a maximum grade of 3 per cent., was built by the Spring Canyon Coal Co. and later turned over to the Denver & Rio Grande for operation.

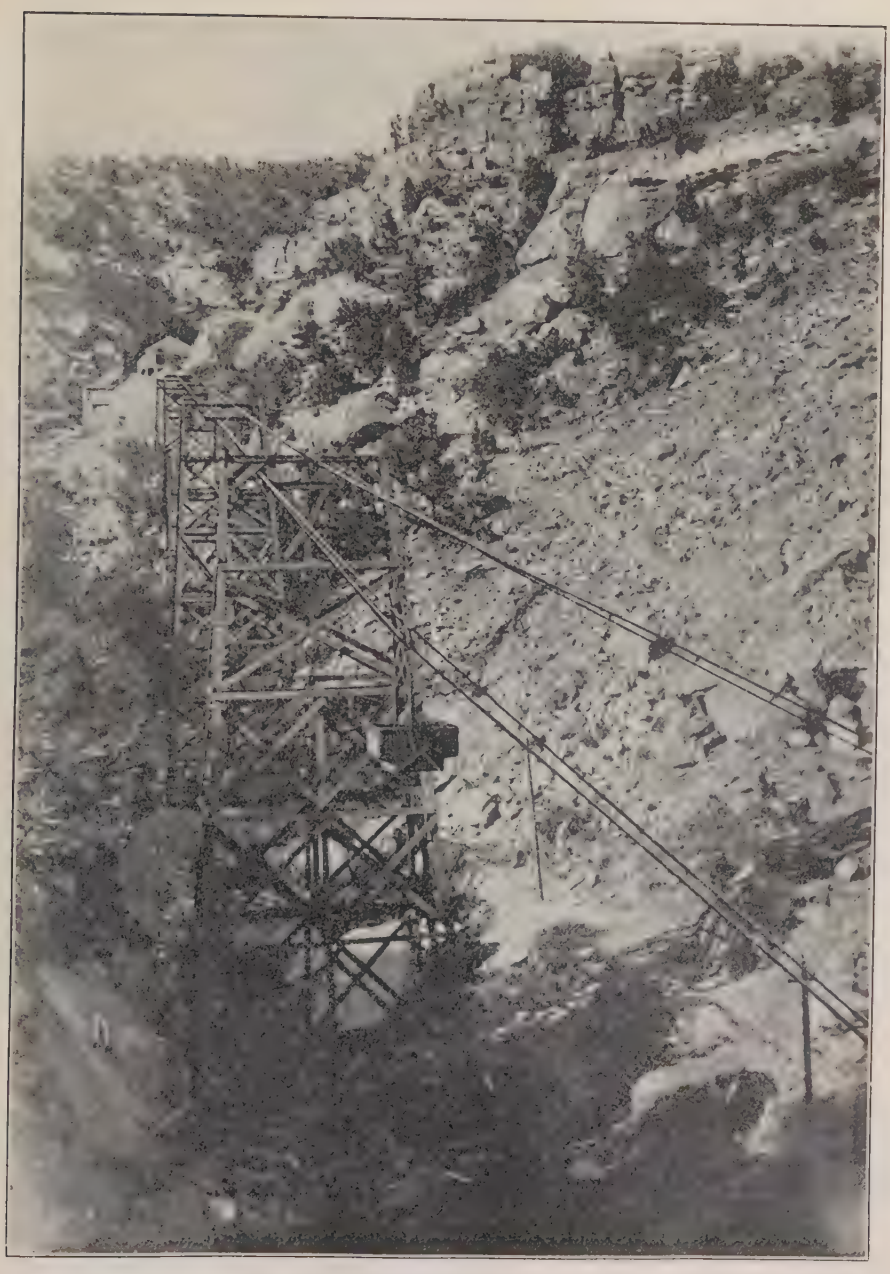


FIG. 2.—GENERAL VIEW OF COUNTRY WHERE TRAMWAY WAS CONSTRUCTED.

The Spring Canyon Coal Co. was organized in 1912 by Jesse Knight, of Provo, Utah, the coal property consisting of about 2,000 acres of land, upon which have been erected 63 rock dwellings and 10 frame dwellings for the convenience of the employees. These houses are modern throughout, being equipped with bath, toilet, electric lights, etc. (Fig. 1.)

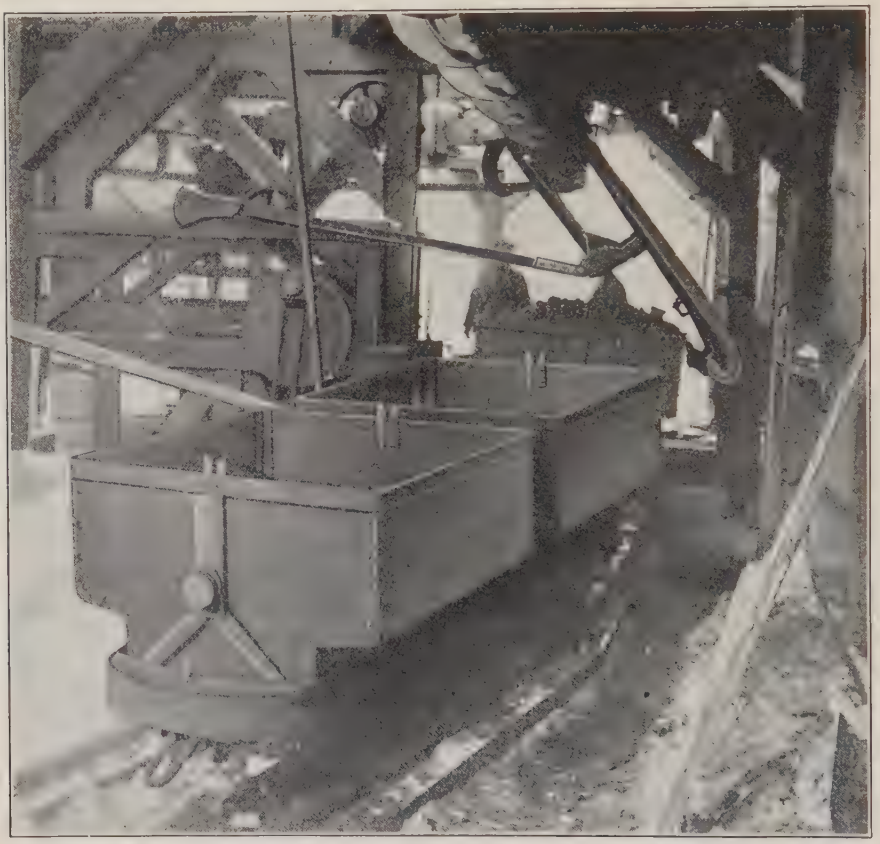


FIG. 3.—EMPTY BUCKETS ON TRUCK COASTING AWAY FROM TRANSFER AND LOADED BUCKETS APPROACHING. HANGERS PUSHED OUT TO CLEAR BUCKETS.

A large rock hotel has been erected with well-furnished rooms and an up-to-date culinary department. The store is of rock also and is conveniently situated in the midst of the greater number of cottages and away from the dust of the tipple and railroad yards.

The company office is in the second story of the store building and is a large roomy office with all the necessary equipment for an up-to-date mine office.

The power plant is located at the terminus of the railroad and near the

tipple, this point being approximately 321 ft. lower in elevation than the portal of the mine some 3,000 ft. away.

The power house is built of rock with asbestos shingles carried upon steel trusses, making the building fireproof. Coal for steam is brought from the tipple, situated nearby, by a belt conveyor and dumped into a bin directly in front of the boilers.

Steam is generated in five Scotch marine boilers of 150 h.p. each. The power-house equipment consists of two 250 kilovolt-ampere sets and one 50 kilovolt-ampere set. The former are Buckeye engines connected to General Electric generators, while the smaller unit is driven by a Sturtevant vertical engine.

The electrical current generated is 2,300 volts, which in turn is transmitted to the mine and transformed by three motor-generator sets of 100 kw. to 250 volts direct current, which is used in the mine for all purposes, including coal-cutting machines, electric locomotives, fans, pumps, etc.

Three veins have been opened. The lower, known as No. 1, is a vein of about 8 ft. in thickness. The coal is a close-grained, glossy black, hard, bituminous coal, suitable for steam and domestic purposes especially, and on account of its hardness makes the best storage coal on the market to-day. This vein is known as a subvein, being 180 ft lower geologically than any of the other workable veins in the region. The floor is of massive sandstone and the roof is a very hard shale, which does not need any timbering in ordinary entry work and very little in room and pillar work.

The next vein above is about 50 in. in thickness. This vein is approximately 15 ft. above No. 1. The hard shale roof of No. 1 extends up to and forms the floor of No. 2. The coal is very similar to that found in No. 1 but not quite as hard. This vein has not been developed to any great extent, due to the fact that it is too small for the large and specially constructed mine cars in use in the other veins.

The third vein, No. 3, is what is known in the region as the Castle Gate vein and is the most persistent of any of the veins in the locality, extending for a great many miles in all directions.

This vein has a thickness of from 3 to 14 ft. throughout the entire region, but where developed by the Spring Canyon Coal Co. workings, shows a thickness of from 6 to 12 ft., most of the workings being in 8 ft. of clean coal.

Although the general dip of the formation in the vicinity is about 7 per cent. all entries are driven sufficiently across the pitch to make the maximum grade not over 3 per cent., permitting the use of electric locomotives both on the main haulageway and cross entires.

Two 15-ton General Electric locomotives are used on the main haulage, and three 6-ton gathering locomotives of the same type are used in connection with the horses in the cross entries and rooms.

The coal from No. 3 mine is brought along the side hill for a distance of 2,500 ft. on a 3 per cent. grade until it reaches the No. 1 vein.

Here it enters No. 1 mine through what is known as the first left entry, and thence is conveyed to the head house, or loading terminal, of the aerial tramway.

Eighteen cars are hauled in each trip by the 15-ton locomotive; 50-lb. rails are used on the main-haulage tracks and 25-lb. on the cross entries.

The motor-generator set referred to above is located in a central position with respect to the haulage system, the greatest distance the trolley line extends in any direction from the motor-generator set being about 2,900 ft.

Heavy copper feed lines are connected to the trolley system at central points in the mine, where the load is heavy, as it is essential to avoid any excessive drop in voltage at the working faces.

The coal-cutting machines in use are of three types and consist of seven Goodman, four Sullivan, and one Jeffrey, all of the short-wall continuous chain cutting variety. These machines are giving excellent service.

The topography of the country precluded the use of a surface incline and the exorbitant price of adjoining property prevented development of grades for motor haulage, or railroad to the mine portal.

The aerial tramway was feasible but had the objection of limiting the output of the mine, due to the small capacity of this system. The tramway was built for 60 tons hourly capacity, to be doubled by additional rolling stock. With the opening of the upper coal measures it was soon evident that this capacity would handicap the mine operations. After much negotiation with the manufacturers, they agreed to develop a capacity of 250 tons hourly on the line.

Our tramway has been so fully described in the *Salt Lake Mining Review* and the *Coal Age*, that the writer will merely outline the principal features, together with such changes as have been made to the original equipment to develop the increased tonnage.

In general, the tramway is 2,900 ft. long and has a fall of 321 ft. between the terminals. The track cables are $1\frac{5}{8}$ and $1\frac{1}{4}$ in. in diameter for loads and empties respectively, working under a tension of 35 and 15 tons each. The running rope is $\frac{7}{8}$ in. in diameter, lang lay. The grips are of the regular friction type, operated directly by weight of load, and hold very effectually.

The line operates at a speed of 500 ft. per minute when driven by motor, and 530 ft. when the motor is acting as a brake to take care of the surplus power developed by the loaded line.

In operation, two buckets are set in a truck frame of 6-in. angles, having a wheel base of 36 in. and an overall length of 11 ft. 3 in. The buckets

are of 40 cu. ft. capacity, rated at 2,000 lb. but in actual service averaging about 2,200 lb.

The car with the two buckets is run on to a platform directly under the tram rail, upon which two hangers are positioned waiting for the buckets.

The car is stopped, the hangers swung under the trunnions of the buckets, and the platform lowered, carrying the truck with it, leaving the buckets suspended in the hangers. The hangers are then free to coast out on the cable. Two hangers with empty buckets are then brought

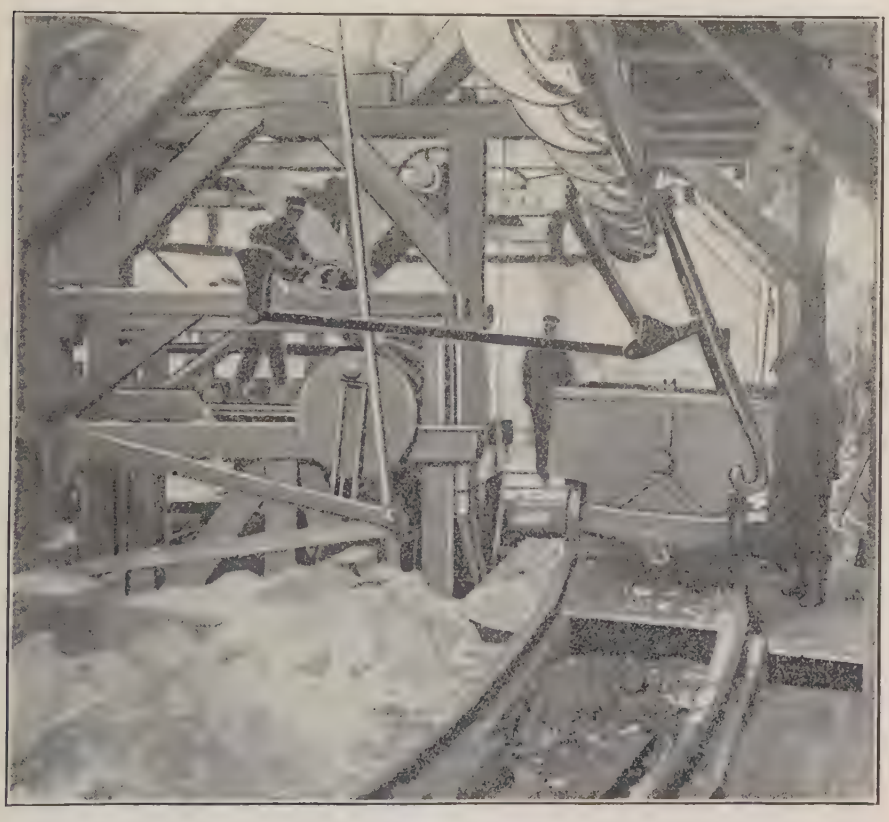


FIG. 4.—PLATFORM IN RAISED POSITION AND HANGERS BEING PUSHED FROM THE BUCKETS.

over the empty truck and the platform raised until the buckets rest on the truck, when the hangers are swung out from under the trunnions of the buckets. (Fig. 4.) The truck with empty buckets is allowed to coast through the kick-back and run into the mine while another load is brought forward for transferring to the hangers.

In developing 250 tons capacity several changes were necessary, as outlined below:

The strength of the cables being based on the weight of the individual

load rather than the total loading of the line necessitated no change. As the line lowers the loads, more power was developed than the original 35-h.p. motor could handle so a 75-h.p. alternating-current motor was installed together with wider belts, clutch, and pulleys. Additional brake sheaves were put in for stopping the line. The speed of the motor is 900 rev. per minute and to reduce it to 22 rev. per minute at the main shaft two jack shafts are used, stepping down the speed three times, twice by belts and once by gearing. To allow proper tensioning of the individual belts adjustable sole plates were provided for the bearings on the countershafts.

Owing to interruptions of power circuits the motor could not be relied upon to control the speed of the line at all times and several expensive runaways were caused by failure of circuit breakers or fuses. To overcome this, the manufacturers worked out a very ingenious control to regulate automatically the speed of line without stoppage in event of any spurt above normal. Briefly stated, this device resembles a large fly ball engine governor and is gear connected to the main shaft to avoid possibility of failure due to belts or clutches. Instead of weights, as in a governor, large vanes are mounted on large pivoted arms which are extended as the speed rises above a predetermined point and effectually prevent the tramway traveling at a higher speed. It is entirely automatic and not affected by changes in temperature. A large dial with a pointer indicating the speed of the line is connected to an electric gong as an additional warning of excessive speed.

With increased loading of the line, the bending of cables over saddles was excessive and a new saddle was developed to overcome this trouble and prolong the life of the cables. (Fig. 5.) The new design consists of two saddles mounted on a 15-in. channel which is pivoted at the center. As the load approaches the tower, the first saddle is depressed and the second is elevated until load reaches the pivot point, when the reverse takes place. The smooth riding of the load over the towers since the installation of these saddles is very marked. To facilitate the turning of the cables with heavy loaded line a new swivel for terminals was designed which allows cables to turn freely under load.

When handling buckets on 60-sec. intervals the coal was dumped into a hopper 9 by 12 ft., but it was found necessary to replace this by one 20 by 22 ft. With this area to dump into there is no trouble in handling buckets on 15-sec. intervals.

Owing to constricted space on the ledge upon which the loading terminal is built the curve of the track for bucket carriers is of short radius. The settlement of part of the structure here caused by proximity of dump fire prevents proper grading of track and causes many derailments of buckets. The buckets derailed here drop several feet. This bends the hangers and breaks small parts and has been the cause of most of our repairs.

The long overhang of the cars is an inherent weakness and a force of men is constantly at work keeping them in proper repair. They are very

FIG. 5.—TYPE OF SUPPORTS FOR CABLE.

clumsy to handle when derailed. Our underground haulage cost is much greater than would be the case with the regular type of pit cars. The

writer believes one bucket per truck would overcome these objections and make a nicer car to handle underground.

The tramway force and wages paid are as follows:

1 foreman at $160 per month	$5.33
3 dumpers at $2.75 per day	8.25
1 bucket spacer at $3 per day	3.00
1 transfer operator at $3	3.00
2 assistants at $2.75	5.50
1 empty-bucket catcher at $2.50	2.50
Total	$27.58

In addition to the above, not actually engaged in handling the tramway, but in handling the cars outside the mine, are:

2 spraggers at $2.50	$5.00
1 weighman at $85 per month	2.83
1 oiler at $2	2.00
	$9.83
Gross total	$37.41

This figure remains unchanged regardless of tonnage, which is unfortunate, as our cost per ton is much greater at 500 tons per day than at 1,500 tons. These costs do not include items of repairs, changes, etc., as the line is in the experimental stage and such costs are borne by the manufacturer.

The manufacturer has shown a commendable spirit of co-operation in the design of various changes and alterations to involve as slight delay in installation as possible and work in harmony with various other machinery, much of which was not furnished by him originally.

Many changes have been made and all seem to be an improvement. Two important drawbacks encountered are: First, the size of the force, whether on large or small tonnage, and second, the trouble encountered with trucks and cost of repairing them. With these points corrected and other improvements made so as to bring the tonnage up to 250 tons per hour, I see no reason why this tramway should not compare favorably with any system of haulage now in vogue in this coal field.

The most coal put over the line so far in a regular day's run has been at the rate of 200 tons per hour for short periods.

Tin and Coal Deposits of the Fu Chuan District, China

BY M. B. YUNG, HONG KONG, CHINA

(Pittsburgh Meeting, October, 1914)

Location

THE Fu Chuan district lies near the eastern boundary of Kwang Si province in southern China and is connected with the coast by the West river and its branches.

The Journey

The first part of the journey from the coast is made very comfortably in one of the West river steamers which runs to Wuchow, the great distributing center for the whole province. From here there are two ways of reaching the Fu Chuan district. One is to drop down the West river for 15 miles and ascend the Yuen river for 120 miles. The other is to ascend the Fu river, which joins the West river at Wuchow, and at Ping Lok take chairs across country. Of the two the former is the more direct and shorter but for other reasons I chose the latter. At Wuchow one must provide his own means of transportation, which may be a native boat anywhere up to 40 ft. in length, but not over 18-in. draft.

The method of working these river boats against the swift current, which in many places develops into rapids, is interesting and worthy of note. Tracking along the bank with the tow line attached to the mast head is the most common procedure, but poling, sculling, sailing, and winding up the rapids by a windlass are used as the case may require.

The boatmen are the hardiest set of fellows I have ever seen, often putting in 15 hr. of the most exhausting labor. The average day's run against the current is 15 miles. On this trip one should have his own food, bedding, and servants, and a guard of 10 or more soldiers is almost a necessity as certain parts of the river are infested with pirates.

The first 60 miles of the Fu river are all through sandstones, above here the metamorphic rocks come in, mica schists and quartzite predominating. In this belt of metamorphic rocks are several alluvial gold areas, both in the river itself and along the banks where the valley widens out, which are worthy of further investigation. Surface samples from some of the big quartzite ridges gave only from 30c. to 50c. per ton in gold.

In this belt of metamorphics, which is about 60 miles wide, the country becomes much more mountainous. Above this belt the river enters the most marvelous and beautiful scenery, caused by the erosion of a

great limestone area leaving its typical topography of spires, domes and fantastic shapes which rise sheer from the clear waters of the stream. Ping Lok, our stopping place, lies in this limestone country. The journey from Hong Kong to Wuchow, a distance of about 300 miles, takes 2 days by steamer; from Wuchow to Ping Lok, a distance of about 150 miles, about 8 days by boat, and from Ping Lok to the mines, a distance of 80 miles, 3 days by chair. Going by the Yuen river from Wuchow the trip may be made in 8 or 9 days.

At Ping Lok the boat was abandoned and we started across country in sedan chairs, the luggage being carried by porters.

The country passed over the first day was all limestone and sandstone but the second day we began paralleling a range of granite mountains which at the end of another day's journey brought us to the tin district, which lies along this granite.

The Nam Shan Granite

At this point I beg to digress for a little to call attention to this granite and to make a tentative proposition with regard to it which I hope, when the opportunity offers, will invite further research.

Extending along the coast line of southwestern Asia from the Yangtze valley to the Malay Peninsula and for a distance of 200 miles inland is a great mountain system covering over 300,000 square miles.

Von Richtofen describes this system in one of his letters on China, giving it the general name of Nam Shan. Owing to its being so broken up and to its absence of any central or dominant range, the existing maps do not recognize it as a system.

Granite forms a very important part of the rocks which go to make up these mountains, especially along the coast of China, where it appears almost continuously in the white barren-looking hills and islands seen from the steamer's deck on the run from Shanghai to Hong Kong. Moreover, this granite does not end with this mountain system proper but appears in Japan projecting beyond its northwestern limit. The mountains along the Inland Sea of Japan are made up of this same granite, which with its white weathering and characteristic topography presents strikingly similar appearance wherever seen.

On the south, the granite continues far into the Malay Peninsula where it plays an important part in the tin deposits of the Malay States.

Proceeding inland from the coast line of China the granite is generally covered by metamorphics and sedimentaries, being itself responsible for much of the metamorphism. For several hundred miles inland, however, wherever erosion has cut deeply, inliers of granite are found, varying in size from a few acres up to many miles in area.

The following are a few common characteristics noted in the various areas visited.

1. The granite is always found underlying older bedded deposits which generally show contact metamorphism.

2. Regional metamorphism is also extensively developed in many places by the granite intrusion.

3. Its composition and texture while varying somewhat are fairly constant in the manner of variation. When the granite is fresh and removed from any contact it has a coarse texture with large phenocrysts of potash feldspar associated with quartz and hornblende and with mica in subordinate quantities. In rare localities the quartz in this granite has apparently segregated out in large masses of many acres in extent. In different localities the amounts of mica vary.

4. Its appearance is white or mottled, more or less speckled with the dark hornblende or mica.

FIG. 1.—IDEAL SECTION OF VALLEY AT WANG KA.

5. Its topography is quite characteristic. In the case of the larger, more prominent mountains, generally found in the interior, where erosion has kept the summits clear of laterite, a massive rounded or dome-like surface is formed with large boulders abundant which weather to a black surface. Where the hills are low and erosion has been unable to keep pace with decomposition, the surface crumbles into a coarse sand which scarcely supports vegetation and gives the characteristic white appearance seen along the coast. This latter phase is by far the most common. In many places one may look for miles over a homogeneous mass of little mountains or hills, which appear like a great collection of ant hills. In this material are found boulders buried in the laterite to a depth of 50 ft. or more, due to concentric weathering of which they are the core.

6. All the tin properties which I have seen or heard of in southeastern China either lie in or contiguous to this granite and I understand that this is generally the case in the Malay States also, so that this granite may be

regarded as the ultimate source of the tin. Furthermore, a considerable percentage of the other mineral deposits within this vast area invaded by the granite owe their origin to the contact of the granite with the sedimentaries.

Upon these general characteristics and upon its geographical and geological distribution, I have felt justified in advancing the opinion that this granite has come from one parent magma.

Certainly its influence upon the geology, mineralogy, and geography of this part of the world has been profound and it is therefore worthy of consideration and study.

The Fu Chuan Tin Deposits

The tin deposits of this district so far discovered are all alluvial and lie scattered over a range of these granite mountains for a distance of more

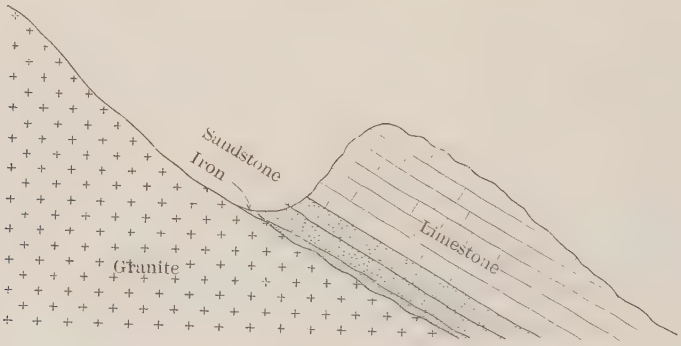


FIG. 2.—SECTION SHOWING TYPICAL CONTACT.

than 30 miles. The average altitude of this range is about 2,500 ft. It is flanked by limestone which forms foot hills and in many places along the contact has been changed to a pure white marble.

This contact of the granite with the limestone, with perhaps a little sandstone intervening, is very generally mineralized with iron existing as both oxide and sulphide. (See Fig. 2.) In one place there is a hill 500 yd. long by 100 yd. wide which is entirely covered with good iron ore in the form of hematite and magnetite.

The tin, which is in the form of cassiterite, is found in the streams which come down from the hills, but most of it comes from the hillsides, where it lies in the cup-like depressions of the limestone and can be easily gained without the removal of much overburden or interference from water. The natives dig it out of these pits, which in extreme cases are 30 ft. deep, but generally much less, and carry it in baskets to the nearest stream, where it is washed in short sluices.

The deposition of the tin high up on these hillsides is evidence of the

erosion that has taken place since the deposit of the tin, there often being a limestone summit between these pockets and the granite from which they came.

There are many level valleys among the foot hills ranging from 300 ft. in width up to a mile. Nothing is being done in these valleys, where one would expect the best ground to be, nor is anything known of their contents. This is due chiefly to the fact that the only parties working in this district are native miners, each working for himself, who depend upon their daily product of concentrate for their daily rice. It is entirely beyond the resources of these people to prospect or work these valley areas, because in the first place cultivated land would have to be bought, and drilling and dredging used to prospect and work the ground, as the ground water level is only a few feet below the surface in most cases. (See Fig. 1.)

The ground worked by the native yields cassiterite from 3 lb. per cubic yard up. It is free from clay and boulders and is ideal ground to wash.

No veins have yet been found, but I was given a piece of cassiterite weighing over $\frac{1}{2}$ lb. which must have come from a vein. No prospecting has been done in the granite, which is not recognized by the natives as a source of the tin, but they have devoted considerable time to looking for pipes in the limestone, which always furnishes them with rich ore; in fact, a very considerable percentage of the tin comes from this class of deposit, which is worthy of description.

Caves are of frequent occurrence in the limestone; these vary from cracks up to caves a mile in length and of variable section from large rooms to spaces through which it is difficult to crawl.

The streams have carried the tin into these caves and deposited it, together with a soft porous lime salt in a more or less stratified deposit. These deposits, which contain from 3 to 5 per cent. cassiterite, are so solid that blasting is resorted to in mining. These deposits are much sought after by the natives and to their minds, not yet mystified by the various theories of ore deposits, answer all the requirements of ore in place.

In one of these caves about 60 men were working. To reach the working place took three-quarters of an hour crawling through passages where cold water and foul air divided the scant space and kept one in constant doubt whether the best chance of seeing daylight again lay in keeping on or turning back.

The Government Monopoly

Up to the present the industry has been conducted as a provincial government monopoly. The government has established stations at the various localities where tin is being washed, where the concentrate is bought from the natives, smelted, and marketed. The government itself does no mining. Any person can start mining without any formality whatever but he is obliged to sell his product to the government station.

The price paid is generally fair, varying from 360 to 450 cash per *cattie*, which is equivalent to 12c. to 16c. gold per pound of 70 per cent. concentrate. This would be a most unreliable figure to count on, however, should a company commence producing tin on a large scale.

Under the new mining laws of China which have just been issued (April, 1914) it is doubtful what will become of this monopoly, since no provision seems to have been made for it. It deserves to pass out of existence as it is undoubtedly holding back the proper development of this promising field.

Smelting

The smelting is done by contract, the smelting contractor guaranteeing 64 catties of tin bullion out of every 100 catties of clean concentrate he receives, and his profit is what he can make over and above this amount. He is also the man who buys the ore from the miners, which serves as a remarkably clever check on the manager of the station.

The ore is bought by inspection, no assays being made, and a certain percentage is deducted for moisture as the ore is always wet when sold.

If the ore is not up to the required standard, it is washed by the smelting contractor, who returns the discard to the miners.

The smelting is done in a small native shaft furnace (Fig. 3). The base of the furnace is a large iron pan and the walls are built up of brick and clay bound with iron. The blast is furnished by a hand bellows made from a hollow log. Charcoal is used in the proportion of 1.3 to 1 of ore and a small charge is added every 10 min. The spout is left open and the blast is not sufficiently trapped so that considerable oxidation of tin takes place at this leak.

Bullion and slag run continuously into a forehearth which is simply a hole dug in the floor. No flux is used and very little slag is formed. It is skimmed from the forehearth and divided into two classes. The top skimmings, which comprise the greater part, are granulated, crushed, and washed, and the concentrate, representing about one-tenth of the lot, is returned to the furnace.

The skimmings coming from next to the bullion are returned direct to the furnace. Thus the two sources of loss are the oxidation of the tin and the discard from washing the slag. The capacity is 700 catties of ore and 300 catties of slag in 12 hr.

The life of a furnace varies from one to three weeks and the cost of rebuilding it is about $10.

The tin bullion generally assays over 98 per cent. and is used by the refiners in Hong Kong to improve the grade of the Yunnan tin. The yearly output is about 150 tons of tin bullion.

Labor is abundant and of excellent quality, the wages averaging about 20c. per day.

The bullion is carried 10 to 20 miles over flat country, 120 miles in 3-ton boats down a shallow river to the West river, where large steamers pass daily. The total cost of transportation from the mines to Hong Kong does not exceed $6.

Should the numerous valleys prove to be dredging ground this district might easily become a very important source of production.

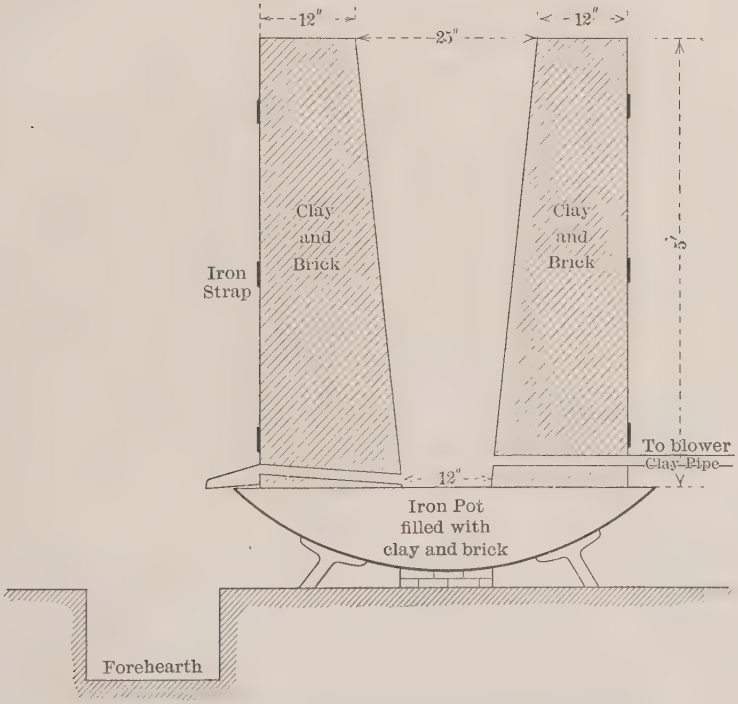


FIG. 3.—CHINESE FURNACE FOR SMELTING TIN.

Fu Chuan Coal

Situated a few miles from the mountains in which the tin is found and right on the Yuen river a fine quality of coking coal is being mined also by the Kwang Si government. Three seams aggregating 24 ft. in thickness occur in a black shale bed not over 75 ft. thick, which in turn lies in limestone. Above the limestone lies a sandstone breccia which has been impregnated with siliceous and iron-bearing solutions, resulting in the iron nodules so often seen about coal outcrops. The surface weathers into a red soil which may be taken as an indicator marking the extent of the coal basin. The coal is developed only to a vertical depth of 160 ft. and a length of 1,000 ft., but judging from various outcrops and from the geology of the surrounding country I should expect to find a coal basin

several miles in extent. The dip of the beds varies from vertical to almost horizontal with the folding of the strata.

The analysis of the coal is as follows:

	Per Cent.
Moisture	1.2
Volatile matter	18.5
Carbon	65.0
Sulphur	0.9
Ash	14.4
Coke	71.0

It burns easily with a long flame and makes an excellent quality of coke.

Transportation is the factor which renders this coal field of small importance at present. The same conditions of transportation exist for the coal as the tin. To exploit this coal on a big scale a railroad would have to be built to the West river, a distance of 120 to 130 miles, but for the present a light railroad from the mine to Ho Yuen, a distance of 12 miles, would cut out the worst part of the river, and from Ho Yuen to the West river boats carrying 30 tons might be used if the narrow channel were deepened in a few places. On the West river barges of any size can be used. I believe this coal could be put into Canton or Hong Kong for a total cost of $3.50 per ton.

Discussion

T. T. Read, New York, N. Y.—I might point out that this paper of Mr. Yung's is on a field that is comparatively little known. It is not as well known as the Kochiu tin field, of Yunnan, which is probably the third in tin output of the world. This field under discussion is considerably further west and the tin which is produced here is, according to Mr. Yung, of somewhat higher grade than the Yunnan tin, and is used by the refiners at Hong Kong to improve the grade of the Yunnan tin. Most of you know that the tin from the Yunnan is shipped to Hong Kong, where it is refined and shipped to Europe. Mr. Yung points out that the Fu Chuan deposits are entirely alluvial and not found in any veins. They differ in that from the Kochiu deposits, which Leclere says are "nullement alluvionels; leur origéne filonienne est de plus evidentes." Granite is the country rock here; doubtless the tin occurred in veins in the granite, and the deposits have resulted from the disintegration of the granite. This is a country of strong erosive agencies. The country is semi-tropical, and the rainfall is heavy, so that one would naturally expect deep surface erosion of this kind.

It is interesting to note in connection with the coal field mentioned that this is only one of many coal areas in China; one of the most fortunate countries in the world, in the wide distribution of its coal. Coal occurs

almost everywhere. It has this disadvantage, however, that it is usually very friable. The strata have generally been subject to disturbance, so that as compared with the coals of this country a great deal of the coal is slack, while our coal contains much lump. The natives have met this difficulty by the general use of the practice of coking. Aside from the big anthracite field in Shansi, the coal throughout the country is mostly bituminous, varying from toward lignite to semi-anthracite. There is very little lignite, except in Manchuria. The Chinese have coked the coal for centuries so that it will stand transportation, since the coal is generally transported in baskets. Fine coal would all filter through, and if they started with a ton, they would arrive with possibly half a ton, so that coke-making is very generally practiced by the natives, in crude furnaces. Now, with the organization of large companies, they have built modern coke ovens. At Pinghsiang there are two large banks of Otto ovens, and the Chinese Engineering & Mining Co. at Tongshan is about to let a contract for building ovens. There are also coke ovens at Penhsihu, in Manchuria. China has tremendous resources in coal which are certainly bound vitally to affect the welfare and the future of the country.

George S. Rice, Pittsburgh, Pa.—I would like to ask if the anthracite deposits in China have been sufficiently exposed to know whether it is a uniform body of anthracite like that in eastern Pennsylvania, or isolated patches such as in Colorado and New Mexico and Washington.

Mr. Read.—Drake has estimated the average workable thickness as 22 ft.; the seams are almost horizontal and occur at no great depth over an area which Von Richthofen estimated at 13,500 square miles.

Mr. Rice.—What is the age of the deposits?

Mr. Read.—Usually they are found in the upper part of the Carboniferous. The Chinese coals average a little higher in the geologic series than our coals in this country. There is a great deal of difficulty in the correlation of strata. But my impression is that they are all rather higher in the scale than our own. They are probably in the upper Carboniferous or Permian. Some of the anthracite coal is the best I ever saw. Some of it is very hard and other varieties are very free burning; and it is fairly cheap. The mining is very cheap; the transportation is what makes it expensive. That is being remedied by the tremendous mileage of railroads which is now being built there. E. W. Parker, in one of his papers, estimated the tonnage of anthracite available for mining as approximately equal to the anthracite resources of the United States.

Mr. Rice.—Is there any volcanic recognized in it?

Mr. Read.—I think not.

Steep Pitch Mining of Thick Coal Veins

BY W. G. WHILDIN, LANSFORD, PA.

(Pittsburgh Meeting, October, 1914)

THIS paper will be confined to a discussion of the methods in use in the property of the Lehigh Coal & Navigation Co. in the Panther Creek valley. Only the methods used in mining the Mammoth vein will be considered, as the methods in use in the smaller veins are adaptations of the same.

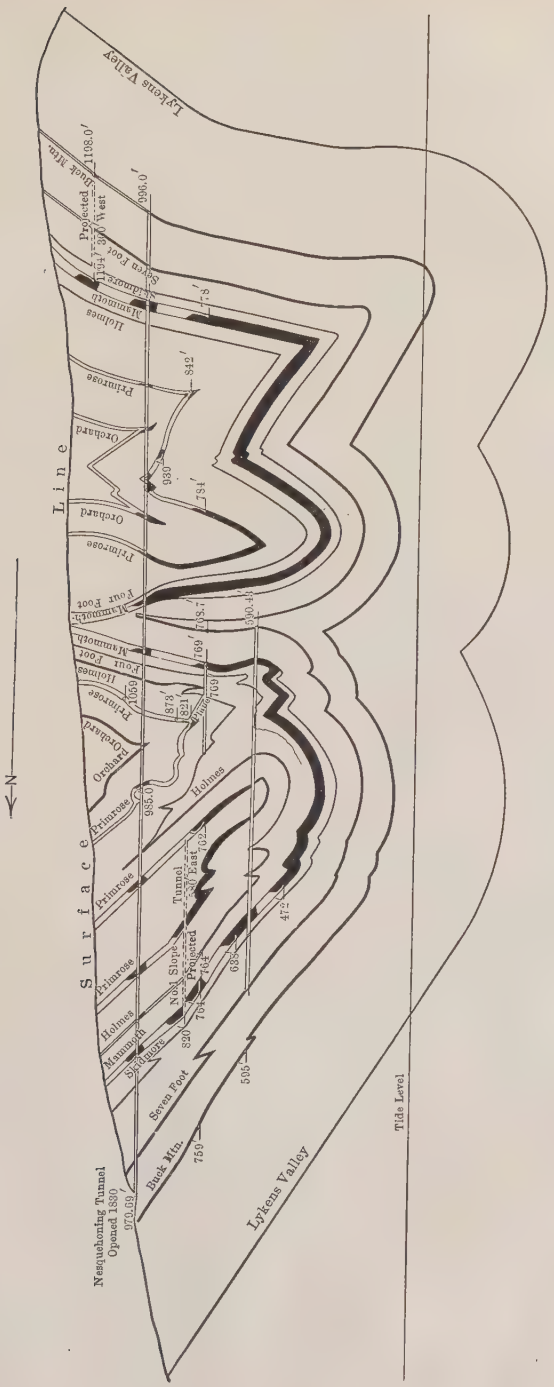
Typical cross-sections of the coal basin are shown in Figs. 1, 2, and 3.

The Mammoth vein varies in thickness. It is 21 ft. at one colliery, 50 to 125 ft. at another, and over 200 ft. at still another. Its normal thickness is 35 to 40 ft.

At some points in the Panther Creek basin the Mammoth vein is made up of three splits—the Bottom, Middle, and Top, while in other portions the dividing strata between the benches amount to only $1\frac{1}{2}$ to $2\frac{1}{2}$ ft., and the vein is practically in one. The section of the Mammoth vein shown in Fig. 4 was taken at the Greenwood colliery, and shows a thickness of 60 ft. The miners know it by benches, as follows: The Three-Foot, the Four-Foot, the Eighteen-Inch Slate, the Bony, the Grey Slate, the Grain Clear, the Five-Foot, the Seven-Foot, the Slaty or Dirty benches, the Blue Slate, and the Top bench.

The present-day methods of mining are an outgrowth of the experiences of the last 100 years in this territory. The first attempt at mining, according to the company's maps, was made at Summit Hill in 1792, when open cuts and pits were made. Later the pillar-and-breast method was introduced. This and other methods will be discussed hereinafter.

Drifts, slopes, and shafts, according to the requirements and physical conditions, are the main openings, from which the gangways are driven. The gangways (haulageways) and airways are driven along the strike of the vein; the gangways in the various veins on the same level being connected by tunnels through the intervening rock. The gangways and airways in thick veins are heavily timbered throughout. The sets of timber, consisting of a collar and two legs, are spaced 5 ft. apart, and later, after the ground has settled, additional sets, called "liners" or "relief sets," are placed between, to relieve the strain on the original timbers, so that the result is a set of timber every $2\frac{1}{2}$ ft.


Surface Line
N
Lykens Valley
Projected Buck Mtn.
1108.0'
Projected Seven Foot West
996.0'
Seven Foot
Mammoth
Holmes
Primrose
Orchard
Four Foot
Mammoth
Holmes
Primrose
Orchard
Primrose
Primrose
Primrose
Holmes
Mammoth
Slope
Seven Foot
Buck Mtn.
Nesquehoning Tunnel Opened 1830
970.0'
Lykens Valley
Tide Level
985.0
Tunnel
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934'
930'
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878
821
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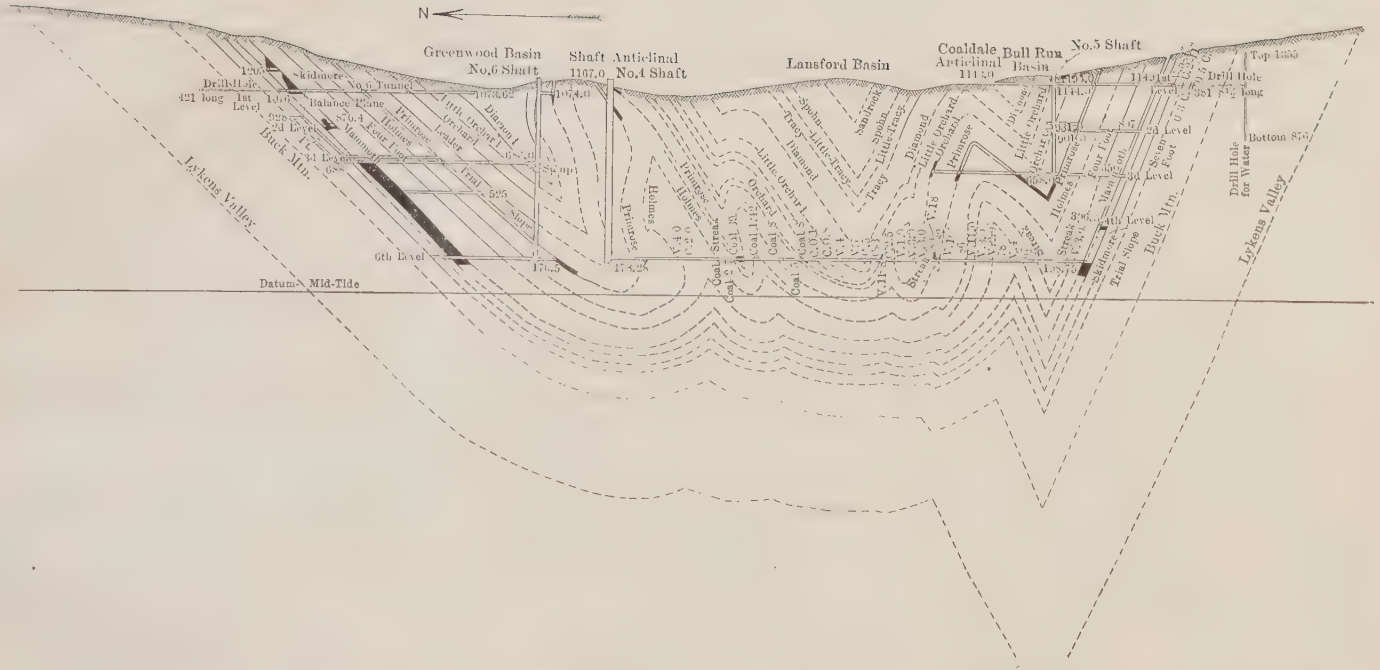
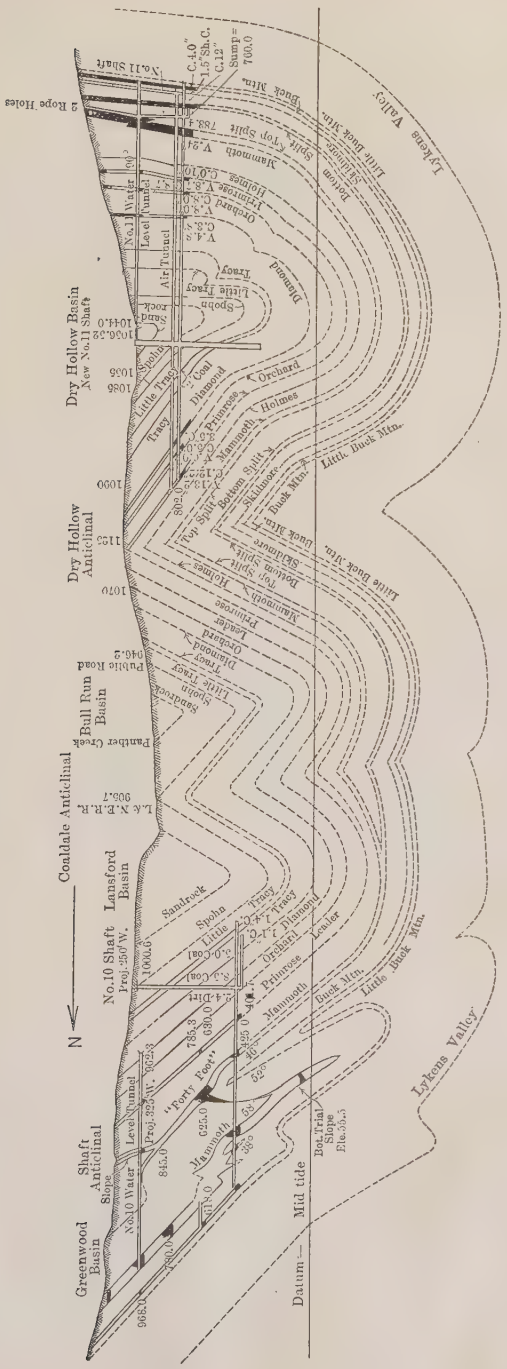


FIG. 2.—CROSS-SECTION THROUGH PANTHER CREEK VALLEY AT LANSFORD COLLIERY ON LINE OF NOS. 4, 5, AND 6 SHAFTS.


Dry Hollow Basin
New No.11 Shaft
Dry Hollow Anticlinal
Bull Run Basin
Panther Creek
Conldale Anticlinal
L.&Y.R.R.R.
No.10 Shaft
Proj. 250' W.
Lansford Basin
Greenwood Basin
Shaft Anticlinal
No.10 Water Level
No.11 Water Level
Air Tunnel
Lykens Valley
Little Buck Mtn.
Buck Mtn.
Mammoth
Primrose
Holmes
Orchard
Diamond
Tracy
Little Tracy
Spohn
Sandrock
Leader
Top Split
Bottom Split
Skidmore
Forty Foot
Mid tide
Datum
Bat. Trial Slope
Ple. 30.3
N
2 Rope Holes
No.11 Shaft
C. 4.0"
C. 12"
Sump
700.0
783.4
1044.0
1036.2
1036
1086
1000
1152
1070
946.2
903.7
1000.0
1050.0
845.0
625.0
621.0
780.0
968.0
735.8
680.0
795.0
Y. 2.4
C. 2.3
C. 0.10
C. 1.13
C. 1.7
C. 0.9
C. 4.0
C. 5.0
C. 3.0
C. 4.0
C. 5.0
C. 1.10
C. 4.0
C. 4.0
C. 4.0

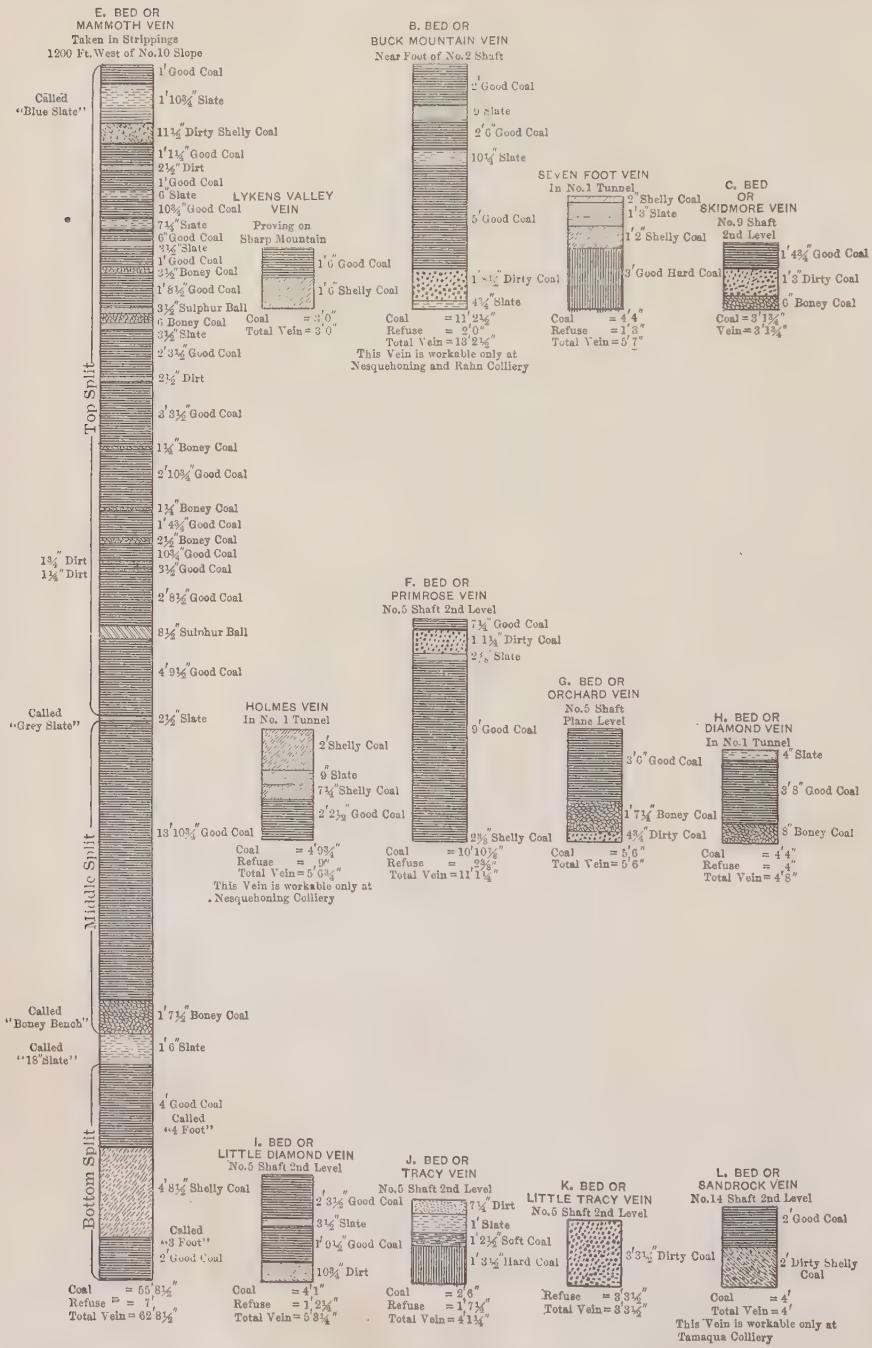


FIG. 4.—SECTIONS OF THE COAL BEDS IN THE PANTHER CREEK VALLEY.

The sizes of the ordinary timbers, when of wood, are: Collar, 8 ft. 6 in. long and 15 in. diameter; legs, each 9 ft. long and 14 in. diameter, set on a batter of 3 in. to the foot. A section of the gangways in the clear, therefore, is 7 ft. 6 in. in width at the top and 11 ft. 9 in. in width at the bottom at the height of the track, and 7 ft. 6 in. high above the track.

The size of the timber in the airways is 10 to 12 in. diameter when of wood, and the size of the opening in the clear is 5 ft. in width at the top and 8 ft. in width at the bottom, and 6 ft. in height. A considerable amount of steel timber is used, the sizes being for gangways a 10-in. I beam, 25 lb. per foot, for collar, and 8-in H columns, 34 lb. per foot, for legs; for airways, a 9-in. I beam, 21 lb. per foot, for collar, and 6-in. H columns, 23.8 lb. per foot, for legs. Some 6-in. H beams, 23.8 lb. per foot, are being tried out for collars instead of the I beams, and so far are giving better results. Where wood is subject to dry rot the use of steel timber has been very successful, but where there is a constant squeezing or heaving it has been found inadvisable to substitute it for wood. A very considerable saving in timbering maintenance has been made up to this time by the use of steel, and its use is being extended.

All turnouts and permanent openings, where timbering is necessary, are being timbered with steel, and sections of various sizes up to 70 lb. per foot are used, according to the requirements. I cannot state at this time what the average life of a set of steel timber in a gangway will be, but it will probably be at least 10 years. The average life of wooden timber in a gangway is three years. In many instances wooden timber has lasted 15 years, and in many others it has lasted less than one year, depending entirely upon the physical conditions. There are also many instances where it was almost physically impossible to hold open certain stretches of gangways and airways until after stee. timber had been installed.

Figuring the average life of wooden timber as three years, and of steel timber as 10 years, the saving made by using steel will be fully 100 per cent.

A notable instance of the saving of steel timber as against wood was in the case of a turnout which had to be retimbered every nine months at a cost of $20 per set for labor and material. The steel timber cost $40 per set for labor and material. Steel timbering was installed four years ago, and will last many years more. It will be seen that the saving amounts to more than $60 per set to the present time.

We use practically all peeled timber, although at times we are unable to procure it. We have not used treated timber in the mines, although some of the other anthracite companies have experimented rather extensively with it for a number of years at one or two of their collieries. We are experimenting with steel timbering instead of wood in a few of our

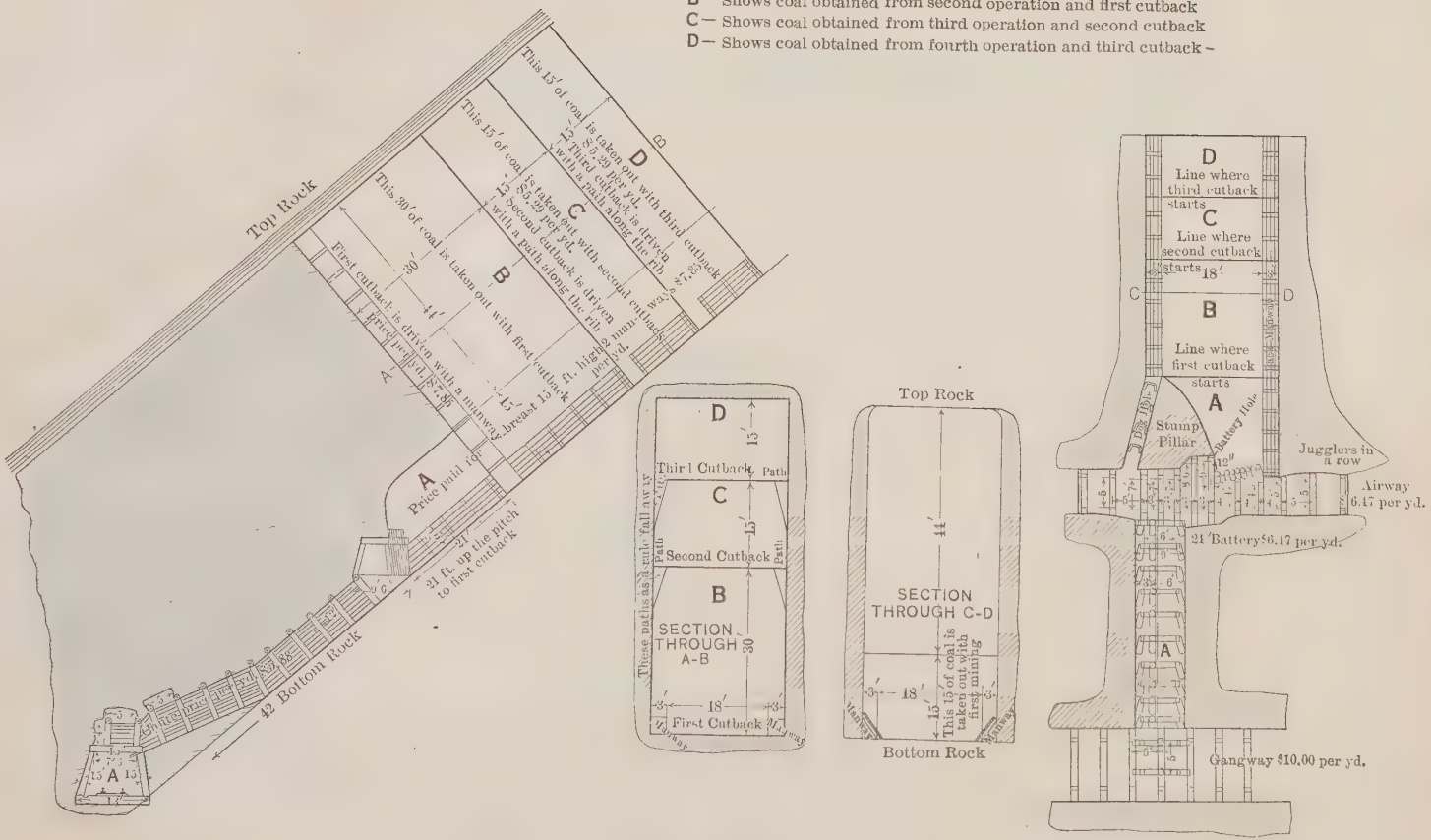

A — Shows coal obtained from first operation
B — Shows coal obtained from second operation and first cutback
C — Shows coal obtained from third operation and second cutback
D — Shows coal obtained from fourth operation and third cutback
Top Rock
42 Bottom Rock
This 30 of coal is taken out with first cutback
First cutback is driven with a manway breast 15
This 15 of coal is taken out with second cutback
with a path along the rib
This 15 of coal is taken out with third cutback
with a path along the rib
21 ft. up the pitch to first cutback
Price paid 15
SECTION THROUGH A-B
Third Cutback
Second Cutback
First Cutback
Path
These paths as rib fall away
SECTION THROUGH C-D
This 15 of coal is taken out with first mining
Bottom Rock
Line where third cutback starts
Line where second cutback starts
Line where first cutback starts
Stump Pillar
Battery Hole
Jugglers in a row
Airway 6.47 per yd.
21 Battery 6.17 per yd.
Gangway $10.00 per yd.

FIG. 5.—CUT-BACK IN NO. 62 BREAST, EAST MAMMOTH VEIN, NO. 10 SHAFT.

main chutes, those which will have to be held open for five or six years, and believe it will result in a saving.

The proper position of gangways and airways with reference to the top and bottom rocks is sometimes a question. In veins pitching from 60° to vertical the gangways and airways are driven along the top rock, for the reason that the loading chutes, on about 30° pitch, can be driven back from the top rock to the bottom rock, providing a safe working place for loaders, and also so that the loose coal can be controlled. In veins with lesser pitches the gangways are driven on the bottom rock and the airway along the top rock.

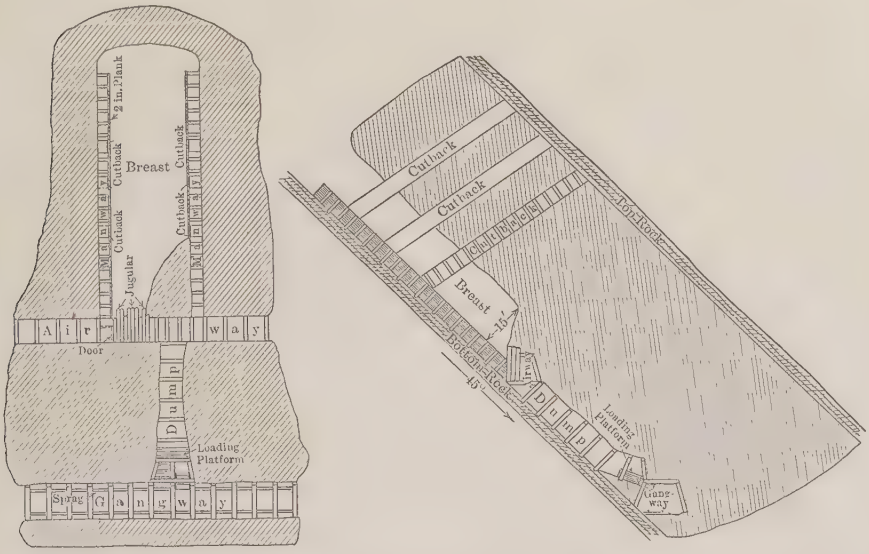


FIG. 6.—METHOD OF WORKING THE MAMMOTH VEIN, LOWER LEVEL, NO. 8 COLLIERY.

The distance between a gangway and its airway, when both are driven along the same rock, varies according to conditions. Years ago the distance varied from 20 to 30 ft., while during the last few years it has been found advantageous in some cases to increase this distance to 50 ft. in the sections where the chutes were driven on 60-ft. centers, resulting in a lower maintenance cost of the gangways.

The method of opening a breast is as follows (see Figs. 5 and 6): Chutes, 6 by 6 ft., partitioned off to make a traveling way and a loading chute, are driven from the gangway on a pitch of 22° until the bottom rock is met, and then up the pitch of the bottom rock, to the height at which it has been decided to have the stump heading or bottom breast cross-cut. At this point a juggler battery is put in and a 4 by 6 ft. cross-cut is driven connecting the top of the chute with the top of the next chute. This

cross-cut is used for ventilating and travel, and is locally known as the "bottom-breast cross-cut."

In forming a breast, the "front" manway or dog hole is driven on one side 20 ft. up the pitch and a back manway carried on the other side; both these are opened off the bottom-breast cross-cut. The opening between the back manway and the battery collar is 10 ft. wide and 8 ft. high, and this opening is carried up the pitch and gradually widened until it reaches the top of the dog hole, where a connection is made, forming what is called a "stump" above the battery. The breast is now 18 ft. wide between the manways. A length of manway is put in on top of the dog hole and the first cut-back is made in the following manner: The top of the dog hole forms the bottom of the cut-back; manways are carried on both sides and the coal is blasted 18 ft. wide between the manways and 10 ft. high all the way back to the top rock of the vein. After the first cut-back has been completed the breast is driven 30 ft. farther up the pitch, where the second cut-back is made. The second and additional cut-backs are similar to the first. Where the pitch of the vein is 45°, for instance, the loose coal does not fill the whole space made. In this case the miners carry a "path" along each rib of the cut-back. Two miners blast the coal, each traveling his own path, until the top rock is reached and all the coal is taken out between the first and second cut-backs. The breast is again driven up the pitch 15 ft., where the third cut-back is made in the same way as the second. If the breast does not run away, or the top coal does not fall, a cut-back is made every 15 ft. up to the old level. In steep pitching veins no paths are necessary, the miners standing on the loose coal to make the cut-backs.

It frequently happens that the breast runs away after the second or third cut-back is made, in which case a chute is driven along the back manway of the breast to a point about 30 or 40 ft. above the face, a battery is put in, and another breast is driven similar to the one just described. If this breast runs away, the chute is carried farther up the pitch past the face of the runaway breast and another breast opened, probably reaching the old level.

At one colliery where the vein is practically vertical and the breasts are driven along the bottom rock, the manways are driven along the top rock, and this has been found of great assistance, because the manways are not so liable to become blocked by falls, thus interfering with the ventilation of the breast.

The driving of breasts as I have just described is the common practice.

Where the vein is divided into its various splits, a different method is used. At the Tamaqua colliery there are a bottom and a middle split, the parting slate being from 12 to 18 ft. thick. The bottom split is 7 ft. thick and the middle split 4½ ft. thick. The gangway is driven in the bottom split, as is also the airway, 40 ft. above. Rock

holes, 50-ft. centers, are driven from the bottom-split airway to the middle split. The middle split is mined by breasts 8 yd. wide, and robbed by skipping the pillars, beginning at the top. After the completion of the robbing, the bottom split is mined in the same way directly underneath the middle-split breasts. The cost of coal mined by this method is high because two breasts have to be driven for 12 ft. thickness of coal.

It was originally planned to use the breast method in the middle split and the chute method in the bottom split; the chutes being driven while the breast was being worked. Care was taken, however, that the robbing at the top of the chutes in the bottom split was not commenced until the robbing had been completed at the top of the breast in the middle split. It was found that as soon as the pillars were cut in the middle split there was so much squeezing and heaving in the chutes in the bottom split that it was almost impossible to hold them open, and the plan had to be abandoned. At the present time, breasts are being driven in both splits satisfactorily, except as to high cost.

Many years ago breasts were driven 10, 12, and 20 yd. wide, and in the old Rhume Run tunnel of Nesquehoning colliery some were driven 30 yd. wide. At the present time breasts are driven 8 yd. wide, and where the vein is free not more than 6 yd. wide. I should say about 40 per cent. of them are 6 yd. wide.

As the width of breasts has decreased, the width of the pillars between breasts has increased. For instance, where 12-yd. breasts were driven on 50-ft. centers, the pillars were 14 ft. wide. At the present time 6-yd. breasts driven on 50-ft. centers give a width of pillar of 32 ft.

In thick veins breast work is preferable to chute work, especially where there is a good supply of air. The amount of blasting necessary is very much less, and consequently the yield of prepared sizes is larger. This is due to the fact that after the first cut-back has been made there is a loose end. Very often a breast will run away after the second or third cut-back has been made, or sooner if the vein is soft, and 2,000 or more cars will be loaded out before the breast has run through to the old lift above. However, when a breast runs it always fills with gas, and remains so until it has run through to the old workings, and this condition requires great care to avoid explosions.

In chute mining in new ground the chutes are driven almost to the old lift above, and as soon as the upper section of the vein has been worked and breaks through to the old workings there is not much further trouble with gas.

The coal is mined by driving chutes back toward the top rock, where breasts are driven first along the top rock, and later up the pitch. Retreat is made down the main chute for 30 to 50 ft. and another chute driven to the top rock, and the operation repeated. This method is carried on until the chute has been robbed down.

All the coal has to be blasted, as heavy falls cannot be expected on account of the shortness of the sections, and also because there are no loose ends. The maintenance cost of the chutes is very high. The result of a test made at one colliery from 1907 to 1909 showed that the breast method cost only 60 per cent. as much as the chute method. However, the recovery by the breast method was only 92 per cent. as much as by the chute method.

The most important question in mining, especially in so far as the ultimate yield of the vein is concerned, is the robbing of the pillars.

In the early mining, the practice was to cut the pillars at the bottom of the breast by stripping a slice off the pillar at the top of the dog hole and bearing into and cutting it through if possible. It was followed up the pitch until it became dangerous to follow any farther, or until the gob from the old breasts rushed in. By this method a large percentage of the pillar was left standing, as the heavy rock from the breast would soon rush in and the place would be abandoned.

A later method was, to drive a chute in the center of the pillar some distance up the pitch, where the pillar was cut through from the rib of one breast to the rib of the other. A larger percentage of coal was obtained by this method, but it was soon found that the maintenance cost was very high, due to the squeezing of the pillars, although it was more successful on the slighter pitches than on the heavy pitches. On the heavy pitch, the coal and rock coming down the chute knocked the timber out, causing heavy maintenance charges.

Another former method was to couple up the old manways of the breasts on each side of the pillar and put in a battery (the old breasts being left full of coal). The bottom benches were taken out between the manways and were cut back in the middle of the pillar toward the top rock, care being taken to leave a small pillar on each side above the old manways which had been coupled, in order not to break them. After these pillar breasts had been driven up to the same height as the face of the old breasts, loading was commenced from the three batteries, trusting that the small pillars over the manways would crush, and all of the coal would be won. This method was found objectionable, as the pillars squeezed so much that it became too dangerous for men to continue to work, and they were seldom able to drive the pillar breast as far as the face of the old breast.

The chute method of robbing pillars was then tried out. These chutes were driven about 25 ft. apart and to within 30 ft. of the old level above, and robbing commenced at the top. It was first thought that by this method it would be easier to reach the coal between the chutes; however, it was soon noticed that so many openings caused such a general squeezing in the chutes that it was almost impossible to keep them open, and in fact many of them were lost. This distance between centers was then

increased to 50 ft., and later to 60 ft. The main chutes were connected up by slant chutes, which were used for ventilation and later for robbing the various blocks of coal in the pillar. This method is the present practice.

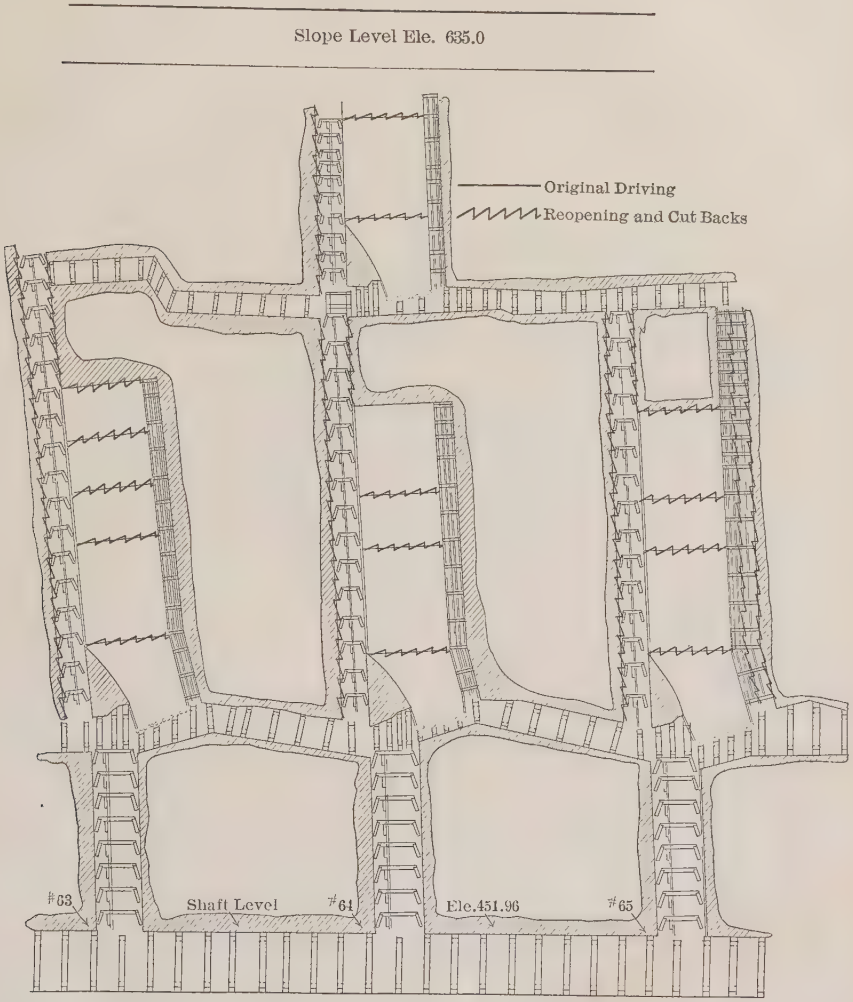


Fig. 7.—Method of Working Breasts and Pillars, East Mammoth Shaft Level, Colliery No. 10.

At the Greenwood colliery a record was kept of every step in the driving of two adjoining breasts and the robbing of the pillar between (see Figs. 7 to 9), in order to determine what recovery of the vein was being made by the method in general use. At this point the vein was 59 ft. thick. The section was 190 ft. long. The total contents of this

section were 590,000 cu. ft. of solid coal, or 8,260 mine cars of 125 cu. ft. each. There were loaded out 5,442 mine cars, a 65.9 per cent. recovery. This work was done in 1909, 1910, 1911, and 1912.

The method of robbing the pillar was as follows: Chutes were

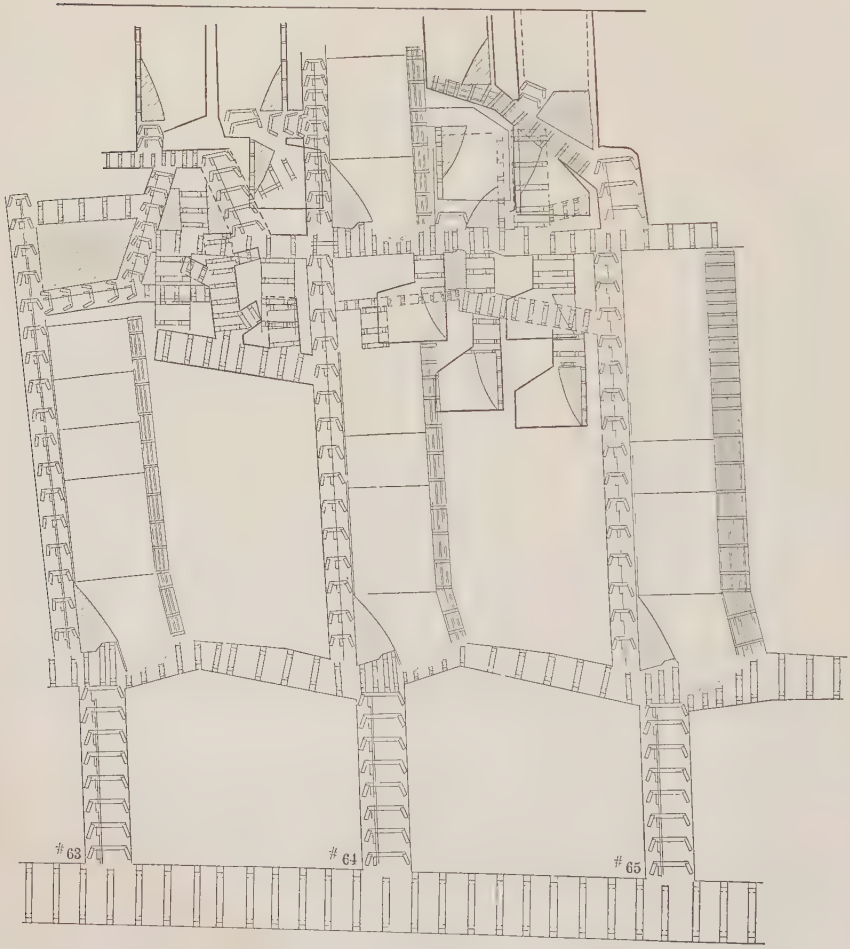


FIG. 8.—METHOD OF WORKING BREASTS AND PILLARS, EAST MAMMOTH SHAFT LEVEL, COLLIERY NO. 10.

first driven along the manways; slant chutes were driven across the pillars, and a chute driven back almost to the top rock. Then a breast was driven up the pitch and the coal mined. Chutes along the small pillars between this breast and the old breasts recovered that part of the vein. After all of the coal had been mined near the top rock, another

LEGEND
Black Original Driving
Purple Reopening and Cutback
Emerald Robbing to Dec. 16, 1910
Deep Chrome " Jan. 17, 1911
Sienna " Feb. 1, 1911
Blue " Apr. 1, 1911
Chrome " May 1, 1911
Carmine " July 1, 1911
Prussian " Sept. 9, 1911
Vermillion " Dec. 1, 1911
Blue " Feb. 28, 1912
Purple " Apr. 1, 1912
Deep Chrome " July 5, 1912
Emerald " Sept. 10, 1912
Sienna " Dec. 10, 1912
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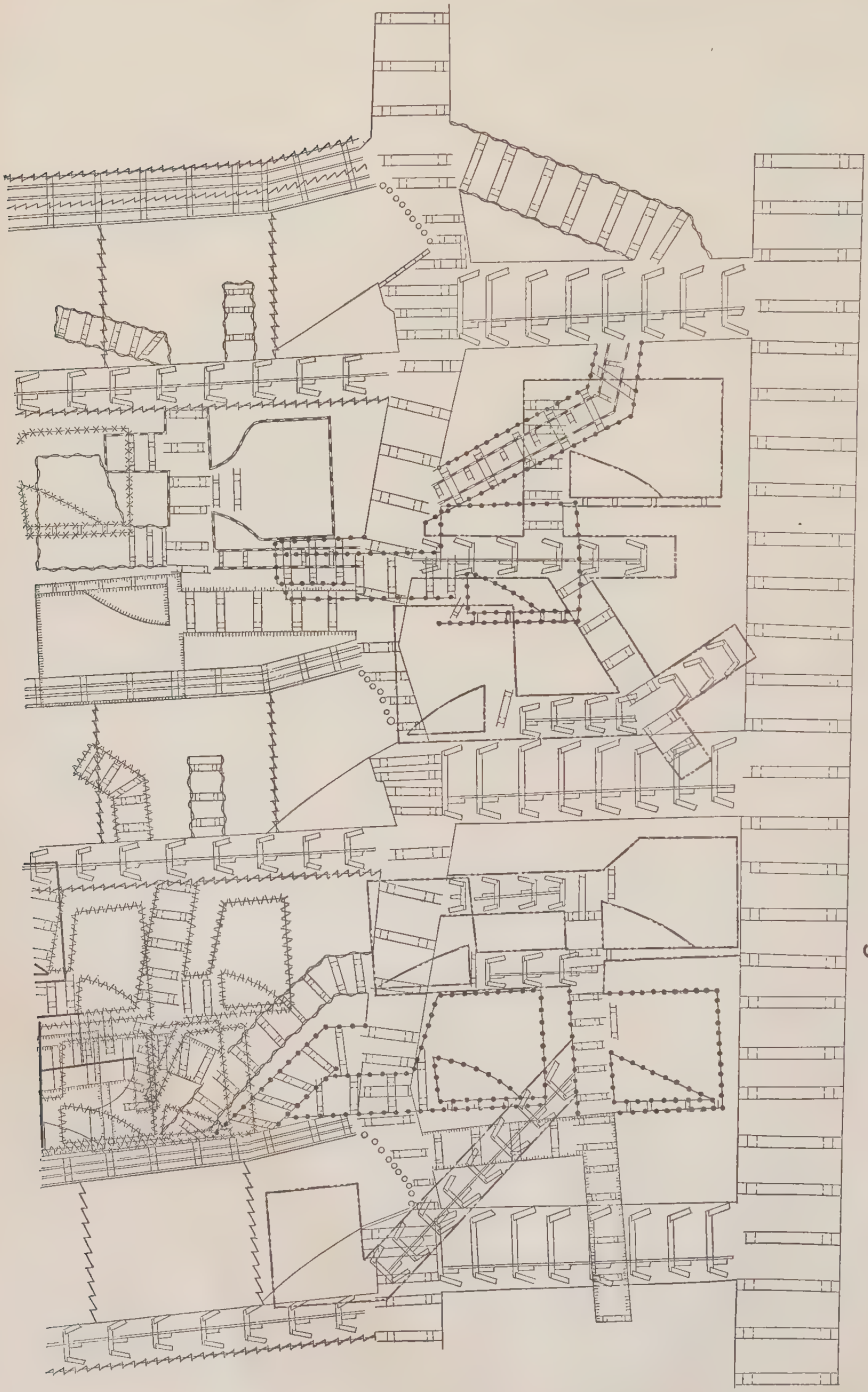


C

FIG. 9.—METHOD OF WORKING BREASTS AND PILLARS, EAST MAMMOTH GANGWAY, SHAFT LEVEL, No. 10 SHAFT, GREENWOOD COLLIERY.

breast was opened off the main chute (driven back through the pillar), and later chutes were driven to mine the small pillars between the breast and the old breasts. When all of the coal possible had been mined in this way toward the top rock, a breast was opened up the pitch along the bottom rock. Again chutes were driven to mine the small pillars between this breast and the old breasts. After all of this had been done the upper part of the pillar was supposed to have been robbed completely; then retreat was made down the chute about 30 ft. and a main chute driven back through the pillar, and the whole operation repeated.

The drawings show a network of chutes and breasts. It has been

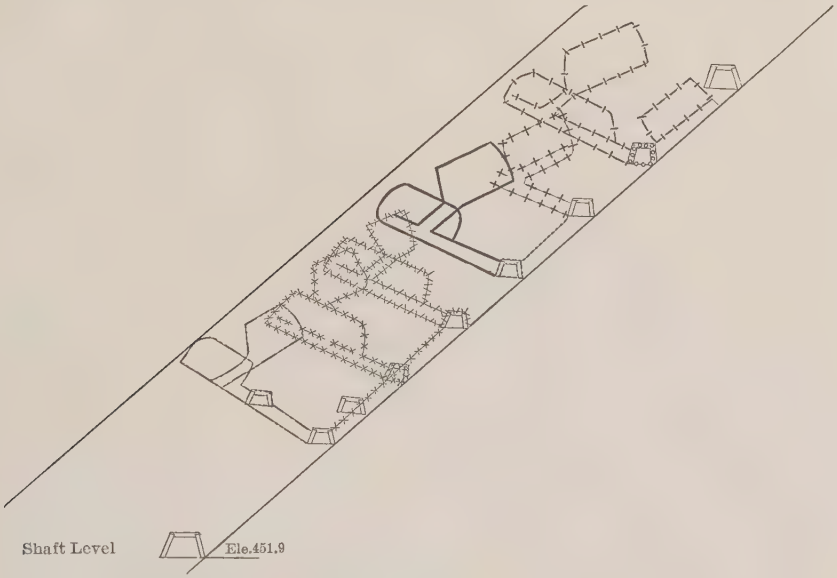
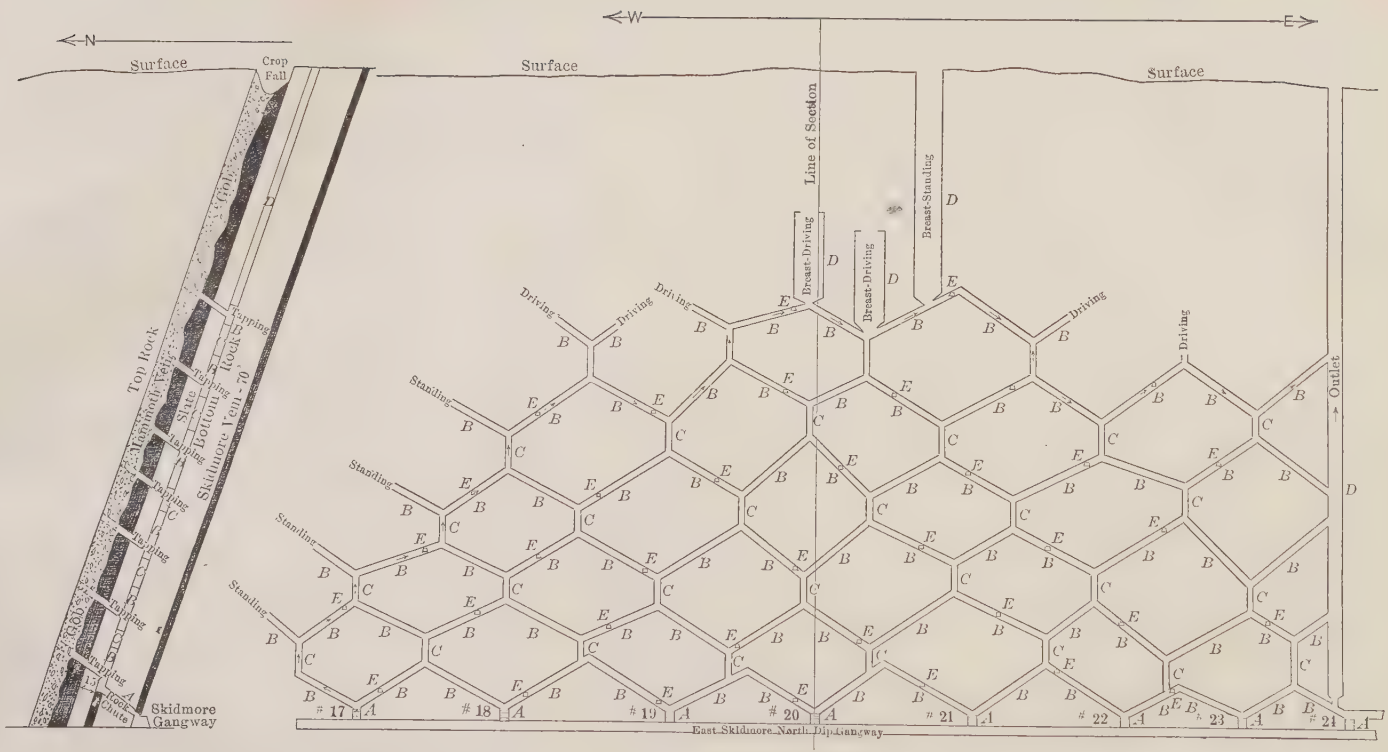


FIG. 9A.—SECTION THROUGH C–D OF FIG. 9.

found by experience that, in order to recover a good percentage of the coal, where the vein is so thick, chutes must be driven in any and all small pillars that have been left between any of the breasts or chutes. In every case, however, the pillar being robbed must be controlled; it cannot be allowed to run away.

The reworking of old territory is always an interesting problem. Years ago it was the practice to reopen an old gangway in the vein, and as a rule the cost was heavy, especially if the ground had not been standing long enough to settle thoroughly. In some instances ground has settled in six or seven years, while other ground has not settled in 15 years. The most satisfactory way to rework the old Mammoth territory has been by driving the gangways in the Skidmore vein and driving rock chutes to the Mammoth vein. At first these rock chutes were driven about



Cross-Section through Chute No. 20.

FIG. 10.—METHOD OF WORKING BOTTOM AND MIDDLE SPLITS OF MAMMOTH VEIN, AS VIEWED AT RIGHT ANGLES TO THE PITCH. NO. 5 SHAFT, SECOND LEVEL, EAST SKIDMORE, NORTH DIP GANGWAY.

(See notes on opposite page.)

100 ft. apart. This distance has now been extended to 120 ft., 140 ft., and even 150 ft. at one colliery. The rock chutes are connected at the top with slant chutes; then, about 10 ft. off the rib of the rock chute, main chutes are driven straight up the pitch, and these main chutes connected by slant chutes across the pitch. These slant chutes provide ventilation and also determine the location of any pillars which had not been worked. At several collieries where the pitch is almost vertical, a straight or box chute about 40 ft. in length is driven up the pitch from the apex or meeting point of the two slant chutes (Fig. 10). At the top of the box chute two other slant chutes are driven, and again at their meeting points a box chute is put in. This is continued and we are able to reach up the pitch any distance desired. In many cases we have driven up the pitch over 400 ft. This method has proved its value and at all times the miners are held safe.

As a rule this system of chutes is made in the bottom bench of the vein, which in most cases has been found solid; the reason being that the coal was extremely hard and could not be mined at a profit many years ago. At intervals, holes are driven through the 18-in. slate into the middle and top benches of the vein, and the gob from the old breasts is drawn out. Where the gob is of such a nature that it can be driven through, chutes are driven on a pitch of 30° to 35° through the vein into the top rock. A pitch of 35° is not too heavy through gob, as more rock has to be handled, and the gob itself is sticky or gummy. Where the bottom bench has been worked, it is necessary to do all of the opening work for robbing purposes in the gob. On the steep pitches we prefer to make the chutes on a pitch of from 30° to 35° and drive across the vein (along the strike) or else directly through it (at right angles to the vein), for the reason that in the gob it is necessary to protect the roof of the chute by driving poles, 6 to 10 ft. in length and 4 to 6 in. in thickness, ahead of the actual working face, to give full protection for the placing of the next set of timber.

It is readily seen that in loose ground, if the chutes were driven directly up the pitch on say 70°, the poling process would be of no benefit, and miners would be in danger from falls of material.

Notes to Fig. 10. (See opposite page.)

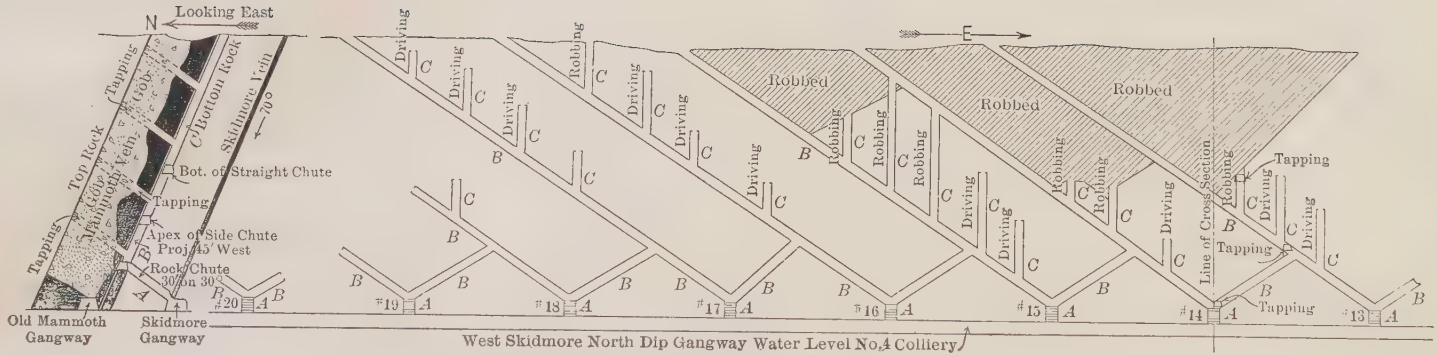
A. Rock Chutes, Skidmore to Bottom Split of Mammoth. About 30 ft. long on 35°, 6 by 8 ft. Chute, 120-ft. centers.

B. Slant Chutes on Bottom Rock of Bottom Split of Mammoth. About 70 or 80 ft. long on 35°, 6 by 6 ft. Chute.

C. Straight Chutes up the Bottom Rock of Bottom Split of Mammoth. About 20 ft. long on 70°, 5 by 6 ft. Chute.

D. Breasts on Bottom Rock of Bottom Split of Mammoth. About 160 ft. long to surface on 70°, 8-yd. breast, 50-ft. centers.

E. Tapping Chutes from Bottom Split to Middle Split of Mammoth. Used as batteries for running coal.



Cross-Section through Chute No. 14.

A. Rock Chutes from Skidmore to Mammoth. 6 by 8 ft. Chute, 35 ft. on 35°, 100-ft. centers.

B. Slant Chutes on Bottom Rock of Mammoth. 7 by 7 ft. Chute on 35°, 300 ft. to surface. Side slants 55 ft. long.

C. Straight Chutes up on Bottom Rock of Mammoth. 5 by 5 ft. Chute, 60 to 100 ft. long on 70°. These Chutes on about 35-ft. centers, depending on condition of gob.

FIG. 11.—METHOD OF WORKING THE MAMMOTH VEIN, AS VIEWED AT RIGHT ANGLES TO THE PITCH. NO. 4 WATER LEVEL, WEST SKIDMORE, NORTH DIP GANGWAY.

In driving rock chutes from the Skidmore to the Mammoth vein, the question often arises as to the economical spacing of the rock chutes, due to the varying thickness of the rock between the two veins. Our experience has been that where the rock is 30 ft. thick the chutes can be driven 100 ft. apart; 40 ft. thick, 120 ft. apart; and 90 ft. thick, 150 ft. apart.

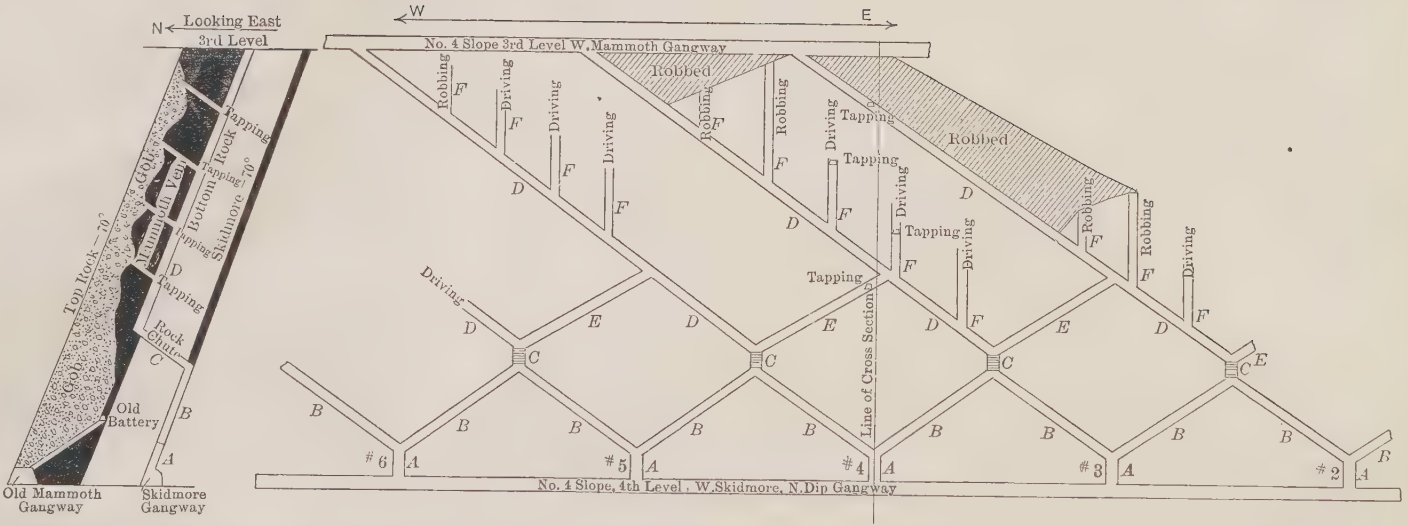
About 1891, when the fourth and fifth levels at the Coal Dale colliery were opened, the gangways were made in the Skidmore vein and rock chutes driven every 50 ft. to the Mammoth vein. These chutes were about 30 ft. in length and on a pitch of 35°. The mining in the Mammoth vein was done according to the usual methods which I have described. These Skidmore gangways required a great deal of repair, due to the very poor top rock of the Skidmore vein at this colliery, and also to the small distance between the chutes.

At one colliery, where the pitch varies between 80° and 90°, a somewhat different system of slant chutes was adopted. The slant chutes were driven across the pitch at about 30° and along the bottom rock. At the junction point of the slant chutes a chute was driven through the vein to the top rock, then turned back again to the bottom rock and slant chutes again driven. There are a number of instances where more than 1,000 ft. of such chutes have been driven from the top of one of these rock chutes.

In other cases (depending upon the conditions), instead of driving the rock chute from the Skidmore to the Mammoth, a chute has been driven up the pitch in the Skidmore to the predetermined height, and then a rock chute driven into the vein, the object being to make the opening above the point where the breast has run away and filled with rock. Where we have been able to work this method it has proved successful.

A variation of this method is being tried out at the Lansford No. 4 colliery (Fig. 11), where the vein is on a pitch of 72°. The rock chutes are driven from the Skidmore gangway to the Mammoth, and from the top of these chutes long slant chutes—7-ft. collar and 7-ft. leg—on a pitch of 35° are driven along the bottom slate of the Mammoth vein to the bottom of the old breaches on the surface. This is being done for the purpose of re-robbing the old workings between the old Water Level gangway and the surface, the lift being about 300 ft. From these slant chutes, straight chutes—5-ft. collar and 5-ft. leg—are driven up the pitch at about 35-ft. centers, depending upon the condition of the gob. The robbing work is started at the top and is continued down the pitch. Great care is taken to keep the robbing faces far enough ahead of the chutes driven later, in order to not bring on a squeeze

Still another variation is being tried out at Lansford No. 4 colliery, fourth or Slope Level. This plan (Fig. 12) differs only from the scheme



Cross-Section through Chute No. 4.

A. Straight Chutes on Bottom Rock of Skidmore Vein. Chute 15 ft. on 70°, 6 by 8 ft. Chute, 140-ft. centers.
B. Slant Chutes on Bottom Rock of Skidmore Vein. Chute 80 ft. on 35°, 6 by 8 ft. Chute.
C. Rock Chutes from Skidmore to Mammoth. Chutes 30 ft. on 35°, 6 by 8 ft. Chute.
D. Slant Chutes on Bottom Rock, Mammoth Vein. Chute 300 ft. to third level on 33°, 7 by 7 ft. Chute.
E. Slant Chutes on Bottom Rock, Mammoth Vein. Chute 80 ft. on 35°, 7 by 7 ft. Chute.
F. Straight Chutes on Bottom Rock of Mammoth Vein. Chutes 60 to 100 ft. on 70°, 5 by 5 ft. Chute, 35-ft. centers.

FIG. 12.—METHOD OF WORKING THE MAMMOTH VEIN, AS VIEWED AT RIGHT ANGLES TO THE PITCH. NO. 4 SLOPE, FOURTH LEVEL, WEST SKIDMORE, NORTH DIP GANGWAY, LANSFORD COLLIERY.

used in the Water Level, just mentioned, in that a 6 by 8 ft. chute is driven from the Skidmore gangway straight up the pitch in the Skidmore vein for 15 ft., these chutes being spaced 140 ft. apart on the gangway. From the top of the straight chutes, slant chutes on a pitch of 35°, 6 by 8 ft. in size, are driven along the bottom rock of the Skidmore vein for about 80 ft. and connected at the top. From that point a rock chute—6 by 8 ft.— on a pitch of 35° is driven back to the Mammoth vein, about 30 ft.; then long slant chutes are driven on 33° pitch along the bottom rock of the Mammoth vein to the old level above. The intention was to strike the Mammoth vein above the batteries of the old breasts which were filled with rock.

We have lately arranged to make further tests of these schemes in

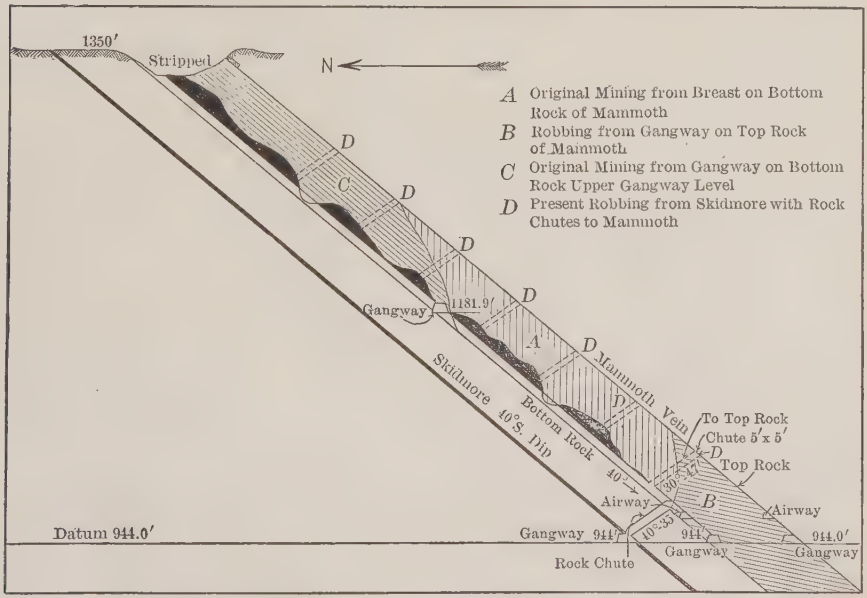


FIG. 13.—CROSS-SECTION SHOWING ORIGINAL MINING AND PRESENT ROBBING OF MAMMOTH VEIN, WATER LEVEL, COAL DALE COLLIERY.

working new territory, in order to make comparisons with the results obtained from the old established piliar-and-breast method.

The question naturally arises, which is the better of these chute plans? The chute yardage driven is practically the same in both methods, with a little saving made in the long slant chute method. The great advantage of the long-chute method over the box-chute method lies in the flow of the coal throughout practically all of its course on the light pitch, causing very much less breakage; the only real drop of the coal being in the straight chute immediately at the robbing point. In the

box-chute method there is bound to be considerable breakage in the straight box chutes unless they are kept full at all times, and this, we know, is frequently neglected by the workmen. At best there is bound to be a certain amount of grinding of the coal in the box chutes.

In many cases the present working is the fourth mining of the same piece of territory. The results of the old mining showed that on the first mining 30 per cent. had been recovered; in the first robbing, 16 per cent.; in the second robbing, 8 per cent.; and in the present working, 10 per cent.

At Coal Dale No. 8 colliery workings, about 60 years ago the first gangway was driven in the Mammoth vein along the bottom rock, and mining was done by the old pillar-and-breast method. After the gangway had been driven to its limit it was robbed out. About 15 years later another gangway was driven through the same territory along the top rock of the vein, and this in turn was robbed out. About 30 years ago another gangway was driven at a level about midway between the two gangways just mentioned and the crop; later the gangway was robbed out. Five years ago, believing there was a large quantity of coal to be won in this territory from the Mammoth vein, we began to drive a gangway in the Skidmore vein 30 ft. underneath the Mammoth and drove rock chutes, 100-ft. centers, to the old Mammoth workings. We figure the present recovery from this section at 10 per cent.

If the present method of mining—that is, mining the Mammoth vein from the Skidmore vein by means of rock chutes—had been adopted in the original mining, there could have been recovered not only the 10 per cent. now being obtained, but also the 54 per cent. recovered by means of the three gangways and their airways above mentioned, with their various breasts and chutes.

The writer had a record kept of the recovery and the cost of working 900 ft. of this territory during the past five years, and estimates that the $45,000 or $50,000 which these old openings cost could have been saved. The cross-section (Fig. 13) of the vein shows the position of the old gangways and the present Skidmore gangway.

In mining very thick veins on steep pitches, it will always be possible, even though the most improved methods are used in the original mining, to win some coal by re-mining, but the cost of obtaining what remains will depend upon market and wage conditions, and may be prohibitive.

Many years ago no coal except that of the very best fracture and best appearance was mined, because it was the only kind found salable. Also, only sizes above chestnut (made over a $\frac{7}{8}$-in. round screen) could be sold. In recent years, all sizes including No. 4 buckwheat (made over a $\frac{1}{16}$-in. round screen) are easily salable, and as a consequence, territory that years ago could not possibly be mined at a profit can now be made to pay.

It has been found of great advantage to make panels of a territory where coal is virgin or where the distance between the Skidmore and the Mammoth veins is too great for practical use of rock holes. The length of these panels depends upon the distance between the veins; the panels in some cases being 600 ft. long, and varying up to 1,200 ft. The paneling system has been very much extended during the last five or six years in the Panther Creek field, and the results have been so satisfactory that the new lifts are being laid out and worked on this plan. At practically all points, none of the veins underlying the Mammoth are workable. The haulageway is being made in the Skidmore vein, which varies from a thin leader to 3 ft. in thickness, and tunnels are being driven at 1,000-ft. intervals from the Skidmore to the Mammoth, Primrose, and Orchard veins, and gangways opened in both directions off the tunnels, thus making a section of about 500 ft. of each gangway which can be mined and robbed. This will allow an increased number of working faces, and also a greatly reduced time through which the gangways must be maintained. In years to come this will result in a great saving in the mining of the Mammoth vein.

Where a panel tunnel is driven a pillar is left over it, this pillar being recovered when the gangway is being robbed out. Every 10 breasts a solid block of coal is left; that is, the coal which would be mined by the eleventh breast is left solid with the exception that the gangway and airway are driven through it. This block is mined on the retreat. The reason is that in case of a mine fire a solid piece is available from which to begin the fight to extinguish the fire. These pillars also have a checking effect on a "creep."

Experience has shown that when the Mammoth gangway, where the vein is 50 to 60 ft. thick or thicker, is being robbed out the best results are obtained by bringing back the face 80 to 120 ft. per year. To many people this has seemed unnecessarily slow, and many attempts have been made to hasten the work, but in every case there has been a very decided loss in the amount of coal recovered. This is due to the disturbance of the thick vein matter causing a general squeeze a considerable distance back from the point of actual robbing and making the maintenance of the gangway not only costly but almost a physical impossibility. In some instances a couple of hundred feet of gangway has been lost, owing to quicker robbing than that mentioned.

When opening a new colliery or a new lift at an old colliery, the question often arises whether it is not better to drive gangways and airways to the boundary limits before opening breasts; also whether it is advisable to open two lifts instead of one. I believe that with the proper system of paneling better results can be attained, because it is possible to have several working faces instead of one, enabling a quicker

development. These panels can then be robbed out in a reasonable time, thus preventing heavy timber maintenance cost.

With the system of driving the gangways to the limit there is a period when nothing but development work can be done, and later nothing but robbing work, whereas in the paneling system some sections can be robbing while others are developing, thus keeping the product of more uniform quality.

The driving of the gangways and airways in the Skidmore vein has not only reduced the timbering expense, but it has improved the ventilation, for the reason that, the vein being so thin, there are no falls to block the airways.

Shot Firing in Coal Mines by Electric Circuit From the Surface

BY GEORGE S. RICE AND H. H. CLARK, PITTSBURGH, PA.

(Pittsburgh Meeting, October, 1914)

WHEN miners in the interior coal fields of the United States began the practice of blasting the coal without undercutting, or what is known as "shooting off the solid," many explosions resulted, some of them of great extent and violence, and attended with large loss of life. One of the means adopted for preventing this large loss of life, though not preventive of explosions, was to employ shot firers to discharge the shots when all the other employees had gone out of the mine at the end of the day's shift. This system still prevails in most of the Central and Southwestern mines.

Following the disaster at Winter Quarters mine, Utah, May 1, 1900, in which 200 lives were lost, a system of firing shots electrically from outside and when the men were out of the mine was adopted at the mines of the Utah Fuel Co.[1] This system has also been used in France in certain mines that are subject to instantaneous outbursts of carbonic acid gas. These outbursts generally occurred at shot-firing time, on account of the blasts releasing the gas held in crevices under high pressure.

The system was also adopted at a number of other mines in Utah, at the Cokedale mine, Colorado, at the Dawson mines, New Mexico, and in some mines of the Tennessee Coal, Iron & Railroad Co. in the Birmingham, Ala., district; and more recently in certain mines in Iowa and Kansas.

At first there was considerable trouble from misfires, and in a few instances from premature firing, while men were in the mine, through the shot-firing lines becoming crossed with power lines. Fortunately, however, no loss of life resulted from such premature firing within the knowledge of the writers, and latterly this danger has been reduced to a minimum by certain measures of precaution. The proportion of misfires, or failures of individual shots to fire, has also been reduced to a very small number through better installation and arrangements of the shot-firing circuits, so that the system has proved very satisfactory, and has stood the test of a number of years' experience. The system and regulations employed in one mine are described in part in *Bulletin No.* 10 of the Bureau of Mines.

[1] Electrical Shot Firing in Coal Mines, *Engineering and Mining Journal*, vol. lxxxviii, No. 5, p. 243 (Jan. 30, 1909).

The need of the adoption of the system, under the conditions which prevail where coal is shot off the solid by black powder or dynamite, has recently been brought to the front by the large number of shot firers that have been killed, particularly in the Southwestern mines.

In some districts where there are restrictions on the quantity of black powder that may be used, and where a careful inspection is made of the holes before the shots are charged, there has been reasonable freedom from accidents; that is, only occasionally has there been an explosion, with the loss of a shot firer or two. This has been the situation in coal-mining States like Illinois and Iowa, but in Indiana, where larger charges of explosives are permitted by law, explosions resulting in the deaths of shot firers have been more frequent. In Kansas and Oklahoma, however, it is admitted on all sides that the methods of shot preparation have been extremely reckless, so that explosions have been very frequent. This has been due to the use of long drill holes, 6, 8 and even 10 ft. or more in depth, drilled straight into the solid, and sometimes loaded with both dynamite and black powder, such shots being used at the face of entries to crack and pulverize the coal in order that it may be easy to make a shearing. So many shot firers in this Southwestern district have been killed and with such regularity, that at some mines they have to be paid $10 per day for a shift of only 2 or 3 hr., and this price is none too high, as the risk is tremendous. One mine, for example, had three shot firers killed in as many explosions in one year and another two in a single month. Many of these explosions, particularly in the pitching beds of McAlester, Okla., which are often dusty and gaseous, are violent, resulting sometimes in wrecking the whole mine. As most of the mines in Oklahoma are on lands of the Civilized Tribes, who are the wards of the nation, the supervision of the leases is under the charge of the Indian Bureau, the U. S. Bureau of Mines acting in an advisory capacity; both bureaus are in the Department of the Interior.

In order to lessen the loss of life in the Indian lands from the use, or misuse, of explosives, and to lessen the number of mine fires and explosions caused by the use of black powder and dynamite, the Secretary of the Interior issued an order effective Aug. 1, 1914, afterward suspended to Jan. 1, 1915, as follows:

"On and after Aug. 1, 1914, except as hereinbefore provided, only such explosives as shall have passed the tests of the (United States) Bureau of Mines, and have been designated as 'permissible explosives,' shall be used in any coal or asphalt mine on the segregated coal and asphalt lands belonging to the Choctaw and Chickasaw Nations in Oklahoma.

"Permissible explosives shall be fired by detonators of a strength not less than No. 6 or otherwise as may be approved by the Bureau of Mines, and shall be used in quantities and under conditions approved by said bureau.

"Other kinds of explosives than 'permissible explosives' may be used in mines in

which the holes are loaded and the shots fired by special shot firers using an electric system from without the mine. The right is reserved to revoke this exception of the use of 'permissible explosives' if it shall be found that explosions of gas and dust are caused thereby resulting in loss of life or of coal through mine explosions or mine fires.

"During the months of June and July, 1914, and subsequently as may prove desirable the Bureau of Mines will supply on application, free of charge, the services of an expert to aid in demonstrating the best methods of using permissible explosives."

At a number of mines on Indian lands where the seam worked is flat, the operators are preparing to introduce undercutting machines and use permissible explosives, which are very effective where the coal is undercut, but which (with exceptions) cannot be used effectively in shooting off the solid, since they are quicker acting than black powder and their charge limit is $1\frac{1}{2}$ lb. In hard or tough coal this is insufficient to throw down the coal without its having been undercut.

In pitching beds, under systems of mining now employed, there are difficulties in introducing the types of mining machines which are most popular. While it is probable that machines of equal efficiency may be developed which will be more suited to the conditions, and also that changes in the system of mining may be made which will permit some of the present semi long-wall or short-wall machines to be used, there are difficulties in agreeing upon labor scales to cover such new work, and therefore some of the operators on the Indian lands, to conform to the Department's order, have indicated their intention of introducing electric systems of shot firing from outside the mine. It is on account of the prospective wider adoption of this system that it seems desirable to have the matter of specifications for the installation of the shot-firing system and the regulations governing it brought before the Institute, with a view to a discussion as to what such specifications and regulations should be, and with a view to making the system as efficient and safe as possible.

Electrical Shot-Firing Systems

Electrical shot firing from the outside of a developed coal mine differs in general from similar shot firing from within a mine only in the number of shots fired at one time, and the greater distance to the shots from the point of firing. There are, however, several differences in the details of the equipment and its arrangement. Outside firing, as compared to inside firing, requires larger firing generators, larger conductors for distributing the current, better insulation on the conductors, and better insulators to support them.

Since the main firing circuit of the outside firing system is a permanent one that extends over a large area and is exposed to various untoward influences and possible mishandling by many people, various switches and safety devices must be used in connection with the system.

It is also essential that there be provided effective means for insuring that no one is in the mine when the shots are fired from the outside.

Factors Contributing to the Success of the System

In order to insure safe and successful operation an outside system must be installed with careful attention to the following points:

1. Freedom from accidental connection with its source of power.
2. Freedom from stray currents from other sources of electricity.
3. Freedom from connection to the earth.
4. Freedom from short circuits between the two sides of the shot-firing line.

Freedom from accidental connection to its source of power requires that several switches located at different points should be connected in series with the line, and that these should be closed in predetermined order, and only by the man authorized to perform this duty.

Freedom from stray currents requires that only properly insulated conductors be used for the shot-firing circuits, and that such circuits be properly installed and protected from roof falls and where crossing or passing near other electric circuits.

Freedom from ground connections is desirable because such connections allow the firing current to be diminished by leakage, and at the same time provide a path by which stray currents may reach the firing circuits.

Freedom from short circuits on the firing line or from even moderate leakage between the two sides of the firing circuit is essential to the proper distribution of current to the detonators or fuses.

Tentative Specification for an Outside Electrical Shot-Firing System

There follows a tentative general specification for the various parts of a shot-firing system, which is subject to change whenever the wisdom of such modification is made manifest as a result of enlarged experience.

(1) *Generator.*—The generator should be a direct-current 250-volt compound-wound machine having a momentary capacity of about 30 kw.; 250 volts is recommended because this voltage will distribute the necessary current throughout the mine without necessitating the use of prohibitively large conductors and it is better than 550 volts because it will produce less strain upon the insulation of the line. However, 550-volt currents have been successfully used, and if there is a source of power of this voltage already installed in the mine power plant, it may be used with some saving in size of shot-firing conductors.

(2) *Distributing Circuit.*—The size of the conductors used should be sufficient to pass at least 1.25 amperes through each room or heading circuit immediately upon closing the firing switch. This current will insure that three or four electric igniters or detonators (if perfect) will explode, even if they vary in resistance 25 per cent. above and below the average. With less current than 1.25 amperes, the Bureau of Mines has found by

experiment that detonators 25 per cent. high in resistance will explode and open the circuit before detonators 25 per cent. low in resistance can become ignited.

(3) The conductors should be insulated with rubber if installed in damp places, but if installed in dry places may be insulated with triple-braid weatherproof insulation. However, as the humidity in coal mines is generally high it is advisable to use the rubber insulation.

(4) The conductors should be supported upon glass or porcelain insulators placed so near together that the wires do not touch anything. It is better not to install the circuits in the same entries with other electric wires, especially trolley lines, but preferably in the parallel entry, and when they have to cross other electric circuits, the shot-firing wires should be completely protected from contact with other circuits both under normal conditions and under conditions that might arise from roof falls and other contingencies. Where a trolley line must be crossed, the shot-firing lines should be protected by special timbering, or placed in a channel cut in the roof, if the roof is low.

(5) The wires that are used for connection between the room switch and the detonators will have to extend for a certain distance beyond the last insulator in order to allow for extensions without making frequent splices. These unsupported ends should be coiled up and hung on the post that supports the last insulator. From that post the wires will, of course, have to extend without support toward the face, but care should be taken that they do not make contact with the floor if it is wet. The ends of the wires should be scraped clean of insulation and the connection with the detonator leads made by cleaning the ends of the leads and twisting them tightly about the ends of the permanent room wires. The detonators in a room should be connected in series but the room and entry circuits should be connected in parallel.

(6) *Switches.*—There should be a double-pole single-throw switch at the entrance to each room or heading of entry. This switch should be mounted securely on a post with the hinges of the switch at the bottom. The room wires should be connected to the hinge or lower terminals of the switch and the power wires to the upper terminals.

(7) There should be at the mouth of each entry a double-pole single-throw 50-ampere switch mounted in the same manner as the room switch.

(8) At the foot of the shaft, slope, or bore hole where the shot-firing circuits enter the mine there should be provided two plugs with flexible lines not less than 5 ft. long to protect further the main circuit of the shot-firing system until all men in the mine have gone out. Provisions should be made for locking the plugs out of circuit.

(9) There should be located in the shot-firer's cabin, in a locked box, a switch that shall be thrown only by the authorized shot firer after all the men are out of the mine and after all other switches have been thrown

in. This switch should be so designed that it can be thrown but once without operating a locked dog, tooth, cam latch, or other restraining device. The switch can be operated again only by unlocking the device. The authorized shot firer shall be the only person having a key to this lock.

(10) There should be located in the power house a switch in a locked box, used for connecting the shot-firing circuit to the generator or power line. This switch shall be thrown in only by the shot firer before closing the shot-firing switch, but not until all the men are out of the mine.

Explanation of Specifications

Specifications (1) to (5) are self explanatory, but it may be advisable to make some further explanation of the paragraphs relating to switches and cutouts.

(6) With reference to the requirement of a switch at the mouth of each room or heading of entry, electrically it is of course not necessary to have such a switch, but to promote safety it is of the utmost importance. The miner in connecting up the shot-firing wires with the detonator or electric igniter leads at the end of his shift when he is about to go out of the mine, would, if there was any stray current through grounding on a power line in the entry, be exposed to the risk of premature firing.

It very much lessens the danger if there is a switch at the mouth of his room attached to a post in plain sight of the inspector or mine official going along the entry; and so placed that the miner is somewhat protected when the switch is thrown. The miner throws this switch in when he goes out of the mouth of his room or heading, after connecting up the shot-firing wires. The room and heading circuits are in parallel so that if by any inadvertence the miner fails to throw in the switch it does not cut out the other working places in the branch entry off which the rooms are turned.

As an additional precaution to prevent premature firing, switches are placed at the mouth of each branch entry, such branch switches being placed in a locked box, the key of which is in possession of the official in charge of the branch or district. This reduces the area affected by possible stray currents until these switches are thrown.

(8) The purpose of putting in the circuit a gap 5 ft. long, as specified above, at the foot of the shaft, slope, or bore hole where the shot-firing circuits enter the mine, is primarily to prevent the possibility of a discharge by lightning, which may strike the outer circuit entering the mine, and would jump an ordinary switch opening. This actually happened in a certain mine, evidently when the men had already left their working places, so that no accident resulted. This plan of using flexible lines 5 ft. long was then adopted at this mine.

(9) The reasons for the specifications for the switch in the shot-firer's cabin and power house are self evident, except that the reason for ar-

ranging the switch so that it may not be again accidentally closed, is to prevent a possible electric flash at the working face in case of grounding of the lines when dust and inflammable gas may have been liberated by the discharge of the shots resulting from the first switch closing.

Checking In-and-Out System

An important part of an electric shot-firing system from outside the mine, is the method of keeping track of the employees entering the mine, to insure that the shot firing is not done until all of them have come out. At many mines, irrespective of the system of shot firing, the checking in-and-out system is used. Where such a system is employed, the check board should be kept in the shot-firer's cabin or adjacent to it.

One excellent method is to have the check board arranged so that every employee and visitor entering the mine may have a number; when the employee is working at the face his number will most conveniently be the same number as used for the checks which he places on mine cars he has loaded. These are generally iron checks; the man checks should be made of brass or other metal to distinguish them from the car checks. When no one is in the mine all of the man checks should be on the board, so that at a glance it can be observed whether or not there is any missing, which would mean one of two things, either that the man was in the mine, or had come out of the mine and gone home without leaving his check. There should be a penalty for non-delivery of the check.

The shot-firer's cabin should be so located that the men in coming out of the mine are compelled to pass the cabin, by means of fences, or by direction of officials stationed at the mouth of the mine. As a man enters the mine he is handed the brass check, and if it is a gaseous mine he may be given his safety lamp at the same time. When he comes out of the mine he gives up his check. When the shot-firing time comes the shot firer examines the board, then if there is any check missing a search is made for the man. There should be accessible for quick reference a book or list showing where each underground employee ordinarily works, if he has a regular station, which is the case with 90 per cent. of the employees.

The checking system, therefore, serves two purposes: One for the protection of the men from shot firing, and the other in case of an accident, such as a fall of roof upon a man working in an isolated place. When he does not appear at the end of the shift he will be sought for. By imposing suitable penalties on those who are careless either in coming out of the mine at the proper time, or in not giving up their checks when they have come out, it is very seldom, at mines where this or similar systems are used, that men fail to give up the check promptly.

When the shot firer has been informed by the underground officials that all the underground main switches have been closed, that the 5-ft.

gap in the circuit where it enters the mine underground has been bridged by the flexible connections, and is sure that all the men are out, he proceeds to the power house, unlocks the switch box and closes the switch that connects the generator to the shot-firing lines; he then returns to the shot-firing cabin, opens the box containing the shot-firing switch and operates that but once. The reason for the mechanical locking of the switch in an open position after throwing, has already been explained.

In mines that use black powder or dynamite for shooting off the solid, fires are frequently started by the explosive, particularly if the mine makes a little inflammable gas at the face. It is, therefore, essential after shot firing that an inspection be made as rapidly as the state of ventilation permits, by a fire boss or other official, both to extinguish incipient fires, and to find out whether there have been any misfires. If there have been some shot failures it will be necessary to re-connect the shots that failed, after first having opened all the branch entry switches and the room switches on those branches where the shots failed. Then after reconnecting the switches leading to these shots, the balance being kept open, the officials come out of the mine and again try to fire those shots. In the event of a second failure it will probably be necessary to examine and test the shot-firing lines for grounds or other defects; and if none are found, to withdraw the explosive charges by the most approved means, and recharge, using fresh detonators or igniters.

Manifestly the success in handling such details will depend upon the efficiency of the mine organization; but as already stated, the percentage of failures in mines where the system has been tried for years is very small.

The object of having all the shots, which may be as many as three or four in a single room or heading, connected in series, is two fold:

(1) To prevent the amount of current falling below that which is necessary to discharge the detonators or igniters.

(2) To insure that either all of the shots in the room or heading be discharged, or none at all. This is because in shooting off the solid where a number of shots are likely to be prepared, especially if there are three or four, the success of some of these shots is dependent upon the success of the others; in other words, they are "dependent shots." In firing shot by shot, the order in which they go off can be assured; but while this undoubtedly is the best system, the next best method is that of having them all go off at once. It is true, it is possible to use delayed detonators or igniters. In this type, pieces of fuse of varying length are inserted, the electric current firing the fuse, and the latter in turn igniting the explosive; thus by regulating the length of fuse the shots may be put off in the order desired. However, the complications of this in the hands of the less experienced or intelligent miners make it somewhat dubious whether there is any material gain over firing simultaneously.

Electric Shot Firing from Outside by Districts or Branch Entries

A number of ingenious systems have been experimented with to fire shots in a mine from outside, sectionally. That is, first one pair of entries, then another pair, and so on, under the theory that this was less severe in its effects upon the mine. However, experience has indicated that the effect of discharging shots throughout the mine at one time is not any more severe than when the shots are fired by sections. In fact, at a large mine it is impossible to tell at the mouth when the shots have been fired, there being so little effect in the way of setting up an air wave. It would seem that the local pressures set up by the shots in the different rooms and entries neutralize one another so that there is no general concussion wave started. Further, there does not appear to be any serious difficulty with the roofs (apart from causing falls of roof), or blowing out of ventilating doors, stoppings, etc., as is liable to take place in the ordinary shot firing, shot by shot.

The expense of a new installation, apart from the generator, is from $1,000, in a mine of moderate size, to about $3,000 in a large mine.

Discussion

Norman V. Breth, Pittsburgh, Pa.—It was my privilege to visit in the last two months on behalf of the U. S. Bureau of Mines 10 mines where electric shot-firing systems controlled from outside the mine are installed.

I will describe briefly what I saw in some of these mines, which are located in Utah, New Mexico, Oklahoma, Alabama, and Kansas.

At a mine in Utah, 230-volt alternating current is used to fire the shots, which is taken from a small transformer in the substation. The shot-firing circuit is independent of all other wires and is tested for grounds every day. There are no lightning arresters on the firing circuit. Porcelain insulators are used inside of the mine, and glass outside.

The switches are situated as follows: One in the substation, one in the shot-firer's cabin, two at each cross entry; where the shot-firing circuit crosses the trolley wire a switch is placed on either side of the entry in series with the shot-firing circuit.

The miners are checked in and out of the mine.

The coal is undercut and is shot down with permissible explosive, the shots being connected in multiple. There are 75 to 110 shots fired each day. A great many times there are no missed shots. If not more than two or three are missed they are left until next day. The fan is kept running at full speed when the shots are fired. The shot-firing switch is closed only once and the whole mine is shot at once.

At another mine in Utah, power wires are used for shot-firing circuits; 500-volt direct current is used to fire the shots. The only sizes of wire used outside of the power wires are Nos. 8 and 14 B. & S.

gauge. The return is connected to the pipe lines and track. Porcelain insulators are used inside of the mine.

The switches are situated as follows: One in power house, the shot-firing switch, one inside the mine controlling the whole mine, one at each cross entry, and one at the end of the trolley wire.

The shot-firing switch is only closed once when the shots are fired. Part of the shots are connected in series and part in multiple. The coal is undercut and shot down with permissible explosives. The miners are checked in and out of the mine. About 150 shots are fired each day. In case of missed shots a new hole is drilled. The whole mine is shot at once. The fan is left running when the shots are fired.

At a third mine in Utah, the power used for firing the shots is taken from a small transformer at 244 volts, alternating current. The main circuit is No. 6 B. & S. R.C., and the room wires are No. 14 B. & S. R.C. Porcelain insulators are used inside of the mine, and glass outside.

The switches are situated as follows: One firing switch, two at each cross entry, and a short-circuiting switch inby from the entry switches. The coal is undercut and is shot down with permissible explosives. The whole mine is shot at once; 200 to 300 shots are fired each day with about 2 per cent. of failures. The missed shots are fired the same night if possible. The shots are connected in multiple. The fan is stopped 5 or 10 min. before the shots are fired and started up immediately after. The men are checked in and out of the mine.

At a fourth mine in Utah, the power for firing the shots is taken from the switchboard at 500 volts, direct current. A pair of power cables is used for the main shot-firing circuit. Nos. 6, 10, and 14 B. & S. R.C. wire is used for the branch circuits. Porcelain insulators are used throughout the mine.

There are two switches at each point where the shot-firing circuit taps the power cables. There is also a shot-firing switch outside the mine. The shots are connected in multiple.

The coal is undercut and is shot down with permissible explosives, the whole mine being shot at once. The booster fan is stopped and the main fan is slowed down before the shots are fired. Missed shots are left until the next day. The men are checked in and out of the mine.

At a mine in New Mexico, the trolley wires are used for one side of the main shot-firing circuits, 250-volt direct current being used for firing the shots. The return is connected to the pipe lines and track. Nos. 8 and 14 B. & S. wire is used for branch circuits. Porcelain insulators are used in the mine.

The switches are situated as follows: One in the substation, one at the entrance to the mine, the shot-firing switch, one at the side track, one at each cross entry, and two at the end of the trolley wire.

The coal is undercut and is shot down with permissible explosives, the whole mine being shot at once. The shots are connected in multiple.

The missed shots are left until the next day. The fan is left running when the shots are fired. The miners are checked in and out of the mine.

At a mine in Alabama the shot-firing circuits are independent of the other wires; 500-volt direct current is used to fire the shots. Nos. 4, 6, 10, and 14 B. & S. R.C. wire is used. Porcelain insulators are used inside the mine, and glass outside.

The main circuit is divided into sections by means of four double-pole switches. The first of these switches inby is provided with a pair of flexible leads 8 ft. long, as a protection against lightning.

The coal is undercut and is shot down with permissible explosives. The detonators are connected in multiple. The whole mine is shot at once. The fan is kept running when the shots are fired. The men are checked in and out of the mine.

At a mine in Kansas, a system, designed by the Scheitzel Shot Firing Machine Co., is used. This system employs a so-called "sparker," which automatically connects to the shot-firing circuit different sections of the mine, one after another. In some parts of the mine the shot-firing circuit is run under the ball of the rail.

The mine is shot in sections, 175-volt direct current being used to fire the shots. The coal is shot from the solid with black powder. The ignitors are connected in multiple. The main circuit is No. 10 B. & S. R.C. The cross-entry circuits are No. 12 R.C. duplex copper; No. 10 iron triple-braid weatherproof is used in the rooms. There are no lightning arresters on the firing circuits. The men are checked in and out of the mine.

At a mine in Oklahoma, the power used for firing the shots is 500-volt direct current. The main circuit is No. 6 B. & S. W.P. and all others are No. 12 B. & S. W.P. Wooden clamps are used instead of insulators. Lightning arresters are installed on the main circuit.

The switches are situated as follows: One in the shot-firer's cabin, one at the entrance to the mine, and one at each cross entry. The shot-firing switch is closed twice when the shots are fired.

The coal is shot from the solid with black powder. All shots are connected in multiple. The missed shots are withdrawn. The fan is shut down 20 min. before the shots are fired. The men are not checked in and out of the mine.

George S. Rice, Pittsburgh, Pa.—One point that I think is particularly commendable is the use of switches in the firing circuits, at the mouth of each room. It strikes me as a most excellent plan, because it permits the inspection of the switches by the foreman who might be traveling along the entry; that was one purpose for its inclusion in the suggested specifications. Another object was also sought, viz.: Where the miner connects up the shots, if he connected them up while the mine trolley and the other power lines were in use, then in case there was any

accidental grounding he might have the shot discharged while still in the room, and possibly receive the direct effect from the blast. I understand this has happened a number of times, but it would not be proper to state where; but by employment of an electric switch on the post at the mouth of the room, or working place, if this is kept open until the miner leaves his room, premature firing is prevented. Further, the position is such that it is plainly observable to any foreman passing along the entry. There is, of course, no necessity of such switches if the mine has no power lines entering it, and provided there are cutouts between the surface and the interior to prevent lightning from entering on the shot-firing wires. The plan of using room switches is also based on the idea that in most mines employing surface shot-firing the miner connects up the detonators with the shot-firing leads. I understand from Mr. O'Brien, General Manager of the Stag Cañon Fuel Co., Dawson, N. M., that the room switches have been taken from these mines; and I think it would be interesting if he would explain why this has been done.

T. H. O'Brien, Dawson, N. M.—Previous to this year the miners loaded their holes and connected the room shooting wires. Some years ago it was reported that on a couple of occasions the shots went off as the connection was being made. That the miners might be better protected, as Mr. Rice has stated, in case of grounding or accidental discharge of the shots while connecting the same, individual switches were installed at the neck of each room. These were closed by the miner on his way out. When he reached his room the following morning, his first duty was to throw out the switch. The foremen, fire bosses, and others made it a practice to see that this was done.

As a further precaution against accidents, the practice of the miners loading their holes was discontinued, and that duty is now performed by special shot loaders, who do not enter the mine until after the shift has been finished and all the men have been checked out.

As soon as the men are out, the power is shut off and the main switch at the mine entrance locked open. At no time is the current turned on while the shot loaders are underground.

The shot loaders take with them sufficient electric detonators for their respective districts. Under no circumstances are they to give or issue a detonator to any one. No pit boss or superintendent has authority to violate this rule.

After the shots have been loaded and connected in the rooms, the shot loader closes the entry switches on his way out. After all the loaders are out of the mine, the main switch is unlocked and closed. The shot loaders then go to the shooting cabin, which is always kept locked, and turn the current on the shooting line.

As soon as the shots have been fired the switches are thrown out and

the shot loaders re-enter the mine for the purpose of inspection. When they reach the entries, the entry switches are thrown out and locked open. They then proceed to the rooms. If a shot has missed the wires are disconnected and a note made of the missed shot. When the inspector returns to the surface the mis-shots are reported in a book kept for that purpose. The following morning, when the fire bosses are making their rounds, they also note any mis-shots and observe whether or not the wires have been disconnected. They also report in their book what they have found in this respect. In this way we have a double check on the mis-shots.

In the case of mis-shots the men are not allowed to go to work in that room on the following day. The reason for keeping them out is because of the possibility of their trying, by some means or other, to fire their shots, not only contrary to the rules, but in violation of the laws of New Mexico.

When any shots have missed, the electrician goes in on the following day and, with suitable instruments, ascertains where the trouble is. This is usually found to be a broken wire, possibly concealed by the insulation.

During the past two years I do not believe we have had more than two detonators which were defective. The shot loaders on the following night reconnect the holes.

It can be readily seen that under the system now in effect the accidental discharge of a shot while the diggers are in their rooms is next to impossible and that the room switches are not a necessity.

C. M. Means, Pittsburgh, Pa.—It is apparent that an electrical system of shot firing from the surface is intended to take care of a condition that cannot be handled in any other practical manner. Such conditions are not met with in the Eastern States and the demand for such systems has been confined to the West and South.

The hazard to human life is lessened in such mines as employ the system, but the danger to property is not lessened and may be increased, because improper shots may be fired in this manner that a shot firer would not risk if fired from the inside of the mine.

A system as described usually represents considerable outlay for the first cost as well as maintenance. It must be properly installed and maintained or it may become a source of danger rather than a safeguard.

The voltage employed may be 250, but owing to the fact that no men are in the mine when power is on the line, there is no danger of accidental contact. The limiting feature is one of insulation and no difficulty should be experienced up to 650 volts. The higher voltage will admit of smaller conductors and should decrease the failures, as current of this higher voltage is more likely to go across dirty or loose connections.

Inasmuch as connections at the face will be made by the miners or shot

firers, it is naturally inferred that many defective or poor connections will be made. The idea of using some sort of a mechanical connector has been considered but does not seem to be a practical solution.

The objection to a shot-firing system of this kind is the increased cost of coal, which may be anywhere between 5 mills and 3c. per ton. In many districts this increased cost represents a large percentage of the margin of profit. The possibility of failures is an item that is not figured, and may still further increase the cost.

The advantage gained is the increased safety to employees.

It is necessary to make a nearly complete analysis of a mine to determine definitely whether or not the introduction of a shot-firing system is the best method of reducing an existing hazard.

Mr. Rice (communication to the Secretary*).—The last remark by Mr. Means touches a most important matter. It was not the thought of the authors of the original paper that the system of electrical shot firing under discussion was the ideal system in all cases; in fact, they frankly acknowledge it is in the nature of a last resort. The idea of firing from the outside of the mine implies there is danger in allowing any one to be in the mine.

In mines which are subject to instantaneous outbursts of gas, either inflammable, or carbon dioxide as in central France, there is no choice, in the interest of safety, to firing the shots from the surface. It will also be universally conceded that it is the safest plan in shaft sinking, or in approaching an underground body of water. Happily, great instantaneous outbursts of gas are practically unknown in this country, so the reason for any extended application of the system arises from danger of coal-dust explosion through blasting coal off the solid with black powder or dynamite, frequently with excessive charges. When the coal can be and is undercut or otherwise freed on the bottom, top, or middle, and permissible explosives within the charge limit are used, there is practically no danger from shots originating explosions and therefore no occasion for firing from the surface so that there need not be any one in the mine. Why the former practice is adhered to is too long a story to take up here, but the fact remains that in a majority of the coal mines in the central West and Southwest, shooting off the solid with excessive charges of long-flame explosives is employed, with a heavy loss of life among the professional shot firers who do the shooting when all the other employees are out of the mine. The record of one Southwestern State shows that the average length of service of a shot firer until he is killed by an explosion originating from a shot that he, or one of his companion shot firers, has ignited is but 2½ years.

There is one feature mentioned in Mr. Breth's description of the meth-

* Received Mar. 7, 1915.

ods used in the mines he visited when shot firing from the surface is employed, that attracts attention, though not strictly related to the subject in hand: namely, the closing down or in some cases slowing down of the fan during shot firing. When a mine is small, it is possibly a good precaution to cut out the fan momentarily to lessen the strain of concussion on the fan, but there does not seem to be any need in large mines such as Mr. Breth examined. On the other hand, if a mine makes any firedamp there is serious danger from the practice. The frequency of explosions in the Southwestern districts, out of all proportion to the output of coal of those districts, would seem to fully refute the claim of lessened danger of explosions by shutting off the ventilating current at shot-firing time.

George S. Rice and H. H. Clark (communication to the Secretary.*)—In the original paper, which presented some tentative ideas regarding certain details of shot firing by electric circuit, the practice of connecting shots in series in any one working place was advocated, primarily on the ground that if one shot depended on the successful blast of the adjacent shot to lighten the burden of the former, or what is known as a "dependent shot," it was important that either all should go or none. Otherwise if a dependent shot was discharged and the relieving shot was not, the burden of the former would be too great and this would probably result in a blown-out shot, with possibility of starting a dust explosion, which would damage the mine. This recommendation was also based upon the fact that if detonators are identical in construction and electrical resistance they can be shot through a considerably greater line resistance when they are connected in series than when they are connected in parallel.

However, experiments made since the paper was written have shown such a tendency on the part of detonators to shoot at different time intervals that the practice of connecting shots in series would probably result in missed shots if the voltage were scarcely enough to cause detonation; or if, as is probable, the application of voltage to the detonators is gradual and not instantaneous, as might be supposed.

At the instant that the firing switch is closed the voltage at those points in the mine that are most distant from the source of power is less, and possibly considerably less, than the voltage that is supplied to the entry nearest to the source of power. The voltage equalizes throughout the mine as the shots are discharged in room after room and entry after entry, but rapid as this process is relatively, it may be sufficiently gradual to cause the more sensitive detonators to fire and the less sensitive detonators to miss if the shots are connected in series. Hence it is believed that to accomplish the purpose sought of minimizing the risk of explosions, the arrangement in parallel is best.

* Received Mar. 10, 1915.

A New Safety Detonating Fuse

BY HARRISON SOUDER, CORNWALL, PA.

(Pittsburgh Meeting, October, 1914)

THE object of this paper is to bring to the notice of engineers a safety detonating fuse by the use of which misfires in blasting may be eliminated and safety in blasting operations promoted.

This new detonator is a French invention and is known as *Cordeau detonant* or detonating fuse and is sold under the name of Cordeau-Bickford. It consists of a lead tube 5 to 6 mm. in diameter, filled with trinitrotoluene.

While applicable to all classes of mining it will appeal especially to those who have to do with deep-hole blasting in open-cut mines or quarries, or any operations where a large number of holes are to be shot at one time.

Since the introduction of the deep-hole blasting in the Cornwall iron mines, much trouble has been experienced from misfires and many experiments were made to determine the best and safest method of blasting.

Before the introduction of the detonating fuse it was found that to insure the best results, two or more high-power electric exploders, not less than No. 6, with specially insulated conductors, should be placed in each hole; the holes wired in parallel, and for a large number of holes the current to be supplied by a live wire to the bus or lead wires at about the middle of the bench. The voltage in general use is 110 to 220, but as our material is a magnetic iron ore and the holes are generally damp, we found it advisable to reduce this to about 30 volts to prevent short circuits. A No. 7 or No. 8 special insulated detonator was adopted. But with the best of care and attention to the minutest detail, we were not able to eliminate occasional misfires and the consequent ever present danger of accidents.

In January, 1914, our attention was called to "Cordeau-Bickford" and we were invited to witness a trial blast with this material at the quarries of the Atlas Portland Cement Co. at Northampton, Pa. This was the first use of this detonating fuse in this country and as it marked a great advance in safety and economy in quarrying operations a brief description of this blast will be of interest.

The rock blasted was a cement limestone shale bench 94 ft. high (perpendicular) with the bottom well cleaned up. There were eleven 5-in. diameter holes, 26 ft. back from the face of the bench and spaced 13

ft. apart, 100 ft. deep and extending 6 ft. below the bottom of the bench. The holes were cleared of water just before loading. The charge was made up of 50 per cent. gelatin and 40 per cent. low-freezing dynamite, with the detonating fuse, 6 mm. in diameter, running to the bottom of each hole.

All the holes were connected with a line of detonating fuse to the end of which a single No. 6 detonating cap was attached. This was fired by a 220-volt current from a live wire. Instantly there was a sharp crack as the fuse exploded, followed by a rumble of the crumbling bank. The rock was thrown well out, leaving a clean wall or face.

The results of the blast were so satisfactory that we immediately ordered 1,000 m. of the detonating fuse for trial in blasting down our ore benches. This was the first application of this fuse to ore mining.

A trial convinced us of the wonderful adaptibility of the detonating fuse to the purpose, and the simplicity and safety in handling brought a sense of relief and satisfaction. We, therefore, immediately adopted it for use in all our deep-hole blasting and stopped the use of the ordinary electric fuse except in shallow holes. We have been using the detonating fuse for five months and in this time we have made a dozen big blasts.

The details of a shot made on Apr. 2, 1914, when 72 holes averaging 30 ft. deep were blasted at one time by means of the detonating fuse are given in Fig. 1. This is a copy of our standard blasting sheet and gives all information in compact form for future reference. Two No. 7 exploders were used and fired by current from live wire.

On May 20, we blasted 26 holes varying from 80 to 85 ft. deep and using in all 7,800 lb. of 40 per cent. nitroglycerin dynamite with the detonating fuse, using two No. 7 exploders discharged by means of a battery. About 38,000 tons of ore were broken down.

Some views of the ore bench before, during, and after the blast are given in Figs. 2, 3, and 4.

The danger arising from the old method of blasting with electric exploders is exemplified by the only too extensive list of accidents due to premature explosions, misfires, etc. Such an accident as occurred during the construction of the Panama Canal in December, 1908, when 22 tons of dynamite exploded prematurely, killing 23 persons and injuring 40 others, would have been practically impossible if this fuse had been used.[1]

In addition to greatly increasing the safety of blasting, the detonating fuse has the additional merit of increasing the efficiency of the explosive charge.

To summarize, we may state that this fuse has three very important qualities, viz.: 1. It is safe. 2. It is instantaneous. 3. It increases the efficiency of an explosive charge.

[1] See *The Panama Gateway*, p. 337.

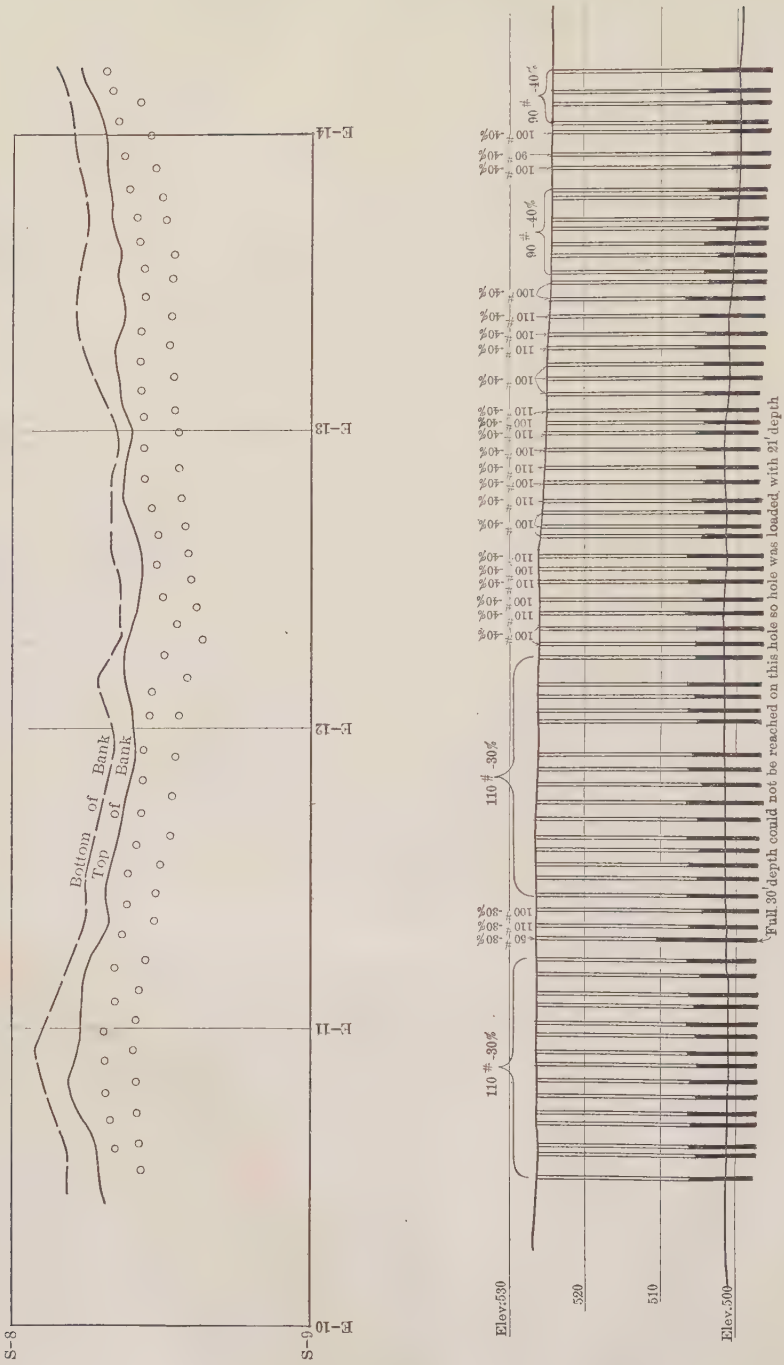


FIG. 1.—BLASTING SHEET OF CORNWALL ORE BANK CO. (See opposite page.)

ORE WELL DRILL SHOT

April 2, 1914

CORNWALL ORE BANK CO.

Shot Fired with Live Wire 2.55 p. m.

Holes were all pumped dry by means of a 2-h.p. gasoline engine and sand pump, thereby letting powder to the bottom of hole.

72 holes fired { 37 holes in front row.
35 holes in back row.

Average depth of holes = 30 ft.

Summary

3,410 lb. 30 per cent.
3,870 lb. 40 per cent.
1,000 M. Cordeau
} $830.07 = Total cost material.

Summary—Continued.

9,129 cu. yd. ore moved or 25,561 tons.

Cost = 0.0909 per yd. or 0.0325 per ton.

Powder used = 0.8 lb. per yd. or 0.28 per ton.

East end of shot consisted of very bony ore and the holes were therefore loaded with 90 lb. 40 per cent., while the rest of the shot was rich heavy ore and 110 lb. 30 per cent. was used or 100 lb. 40 per cent.

All Cordeau and no exploders used in holes.

Holes were all very wet. Majority of them were over half full of water.

Better results were obtained by this shot than by any preceding one. Less powder was used. Bottom came out first.

FIG. 1A.—DESCRIPTIVE DETAIL OF BLAST AS FILED ON BLASTING SHEET OF CORNWALL ORE BANK CO. (See opposite page.)

1. *Safety.*—There is no danger in the handling or storage of the fuse. It cannot be exploded by friction, fire, or ordinary shock. It requires the use of a strong blasting cap properly attached to explode it. In blasting charged holes, the cap or exploder can be applied outside the hole, thus avoiding the danger of burned powder caused by side spit from ordinary fuse; also any risk of accident while tamping and the risk from a portion of an unexploded charge accompanied by a cap remaining in the débris from a blast is entirely obviated.

2. *Speed.*—The average rate of speed of this fuse is estimated to be close to 17,000 ft. per second, so that when it is used the explosive

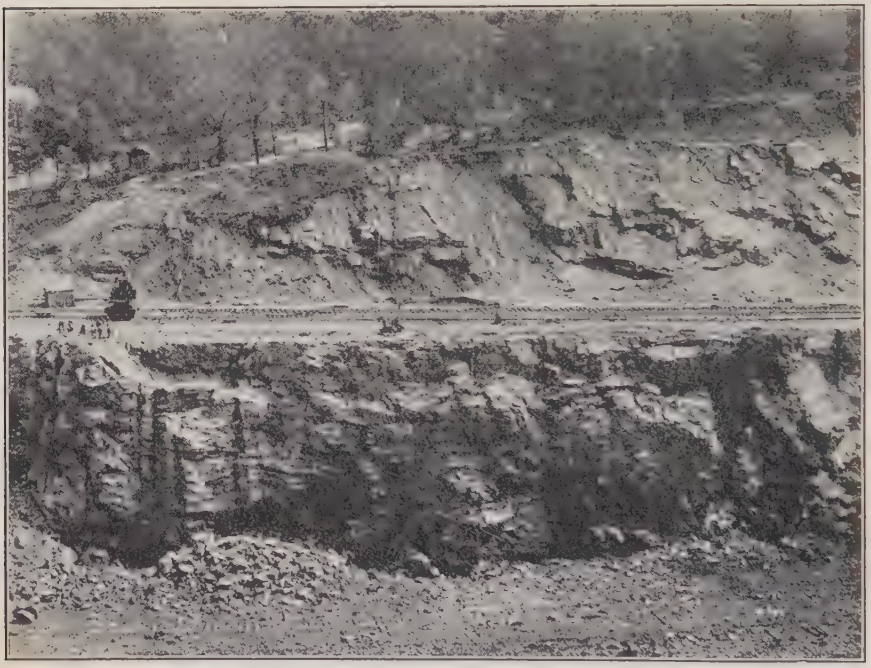


FIG. 2.—LOADING THE HOLES FOR THE BLAST.

charge is detonated instantly throughout its entire length, instead of at one point as is the case with the blasting cap or electric exploder.

3. *Efficiency.*—It is known that the speed of an explosive decreases as the explosive wave travels away from the detonator. That the powder in a hole has the strongest explosive effect around the exploder is evident from an examination of the face of the bank after a shot. This can be demonstrated also by placing sticks of dynamite on the ground end for end, about 6 in. apart, with the cap in the first stick. The explosive force gradually lessens until it finally ceases to progress, leaving the farthest sticks unexploded.

By using this fuse the charge is detonated instantaneously throughout its entire length. This results in a saving of about 10 per cent. of explosives as determined by results obtained at Cornwall and elsewhere. It is not affected by heat, cold, or moisture, and lasts indefinitely without deterioration.

It is wound in continuous lengths on spools containing 100, 200, or 300 ft. each, and weighs about 7 lb. per 100 ft. It is accepted by transportation companies without restrictions except that it shall not be packed with other high explosives.

FIG. 3.—DURING THE BLAST.

The method of applying the fuse is shown clearly by the accompanying sketches and instructions.

Fig. 5 shows how the holes are connected by the detonating fuse. The method of making connections between main and branch lines is shown in Fig. 6, and that of attaching the detonator to the fuse in Fig. 7. Either ordinary fuse and cap, or an electric exploder, can be used to set off the detonating fuse.

How to Attach a Detonator to the Detonating Fuse.—First, be sure the end of the detonating fuse is freshly and squarely cut off. If ordinary fuse is used, attach a cap by crimping it to the fuse in the usual manner. Next attach a union to the detonating fuse by crimping the end opposite

the slit. Insert the cap in the slit end of the union till it comes firmly against the detonating fuse and then slip the ring over it to hold it firmly in place. Be sure the detonating fuse is cut off squarely and that the cap is firmly seated against it. A space of $\frac{1}{8}$ in. between cap and fuse may be sufficient to cause a misfire.

Branch line *A* is joined to trunk line *B* by twisting the split ends, one to the right, the other to the left of the intersection. These joints may be protected from danger of loosening or breaking by winding with

FIG. 4.—AFTER THE BLAST.

electric tape. This also protects them from moisture. This precaution is not necessary except where many connections have to be made or wet work is encountered.

This operation is not difficult, but it is very important, and should be done carefully and in accordance with three simple rules.

1. Remember that it is the *branch* fuse that is to be slit and twisted about the *main* and never the opposite.

2. Be sure to make a firm joint.

3. At the point of intersection, be sure to lead the branch fuse away from the main at a *right angle*, at least for an inch or two. Beyond that curves and angles do no harm.

If the direction of the explosive wave is carefully borne in mind,

there can be as many branches and sub-branches as are desired. To slit the end of the branch line an instrument called a "Cordeau slitter" is provided. It can, however, be done easily with a sharp knife. After slitting it for a distance of 3 or 4 in., separate the legs and, after placing the main snugly in the crotch, wind the legs about it, one to the right and the other to the left. If some of the trinitrotoluene is lost from the split ends, no harm is done provided the main is pushed firmly against the exposed material in the crotch. If many connections have to be made, it is possible that the handling and moving of the fuse may displace some of the connections first made before all can be completed. To guard against such an occurrence it is better to wind each joint with electric tape. This will hold them firmly in place. Before a shot is fired, the line should be thoroughly inspected. See that the branches leave the main at right angles. Be sure that they are all made firm.

On very large and important work where more than one main is em-

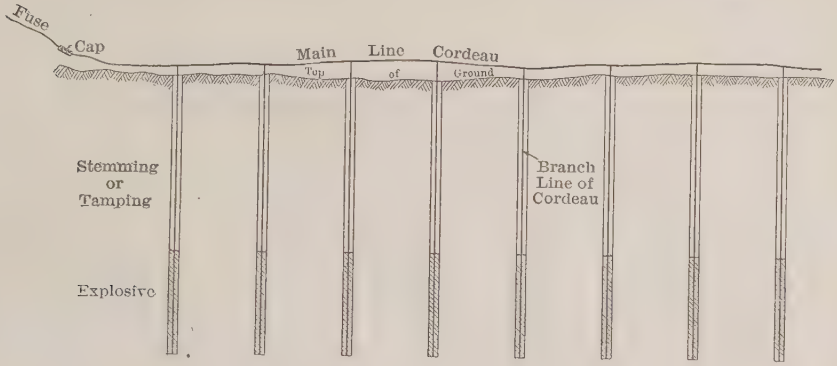


FIG. 5.—HOLES CONNECTED BY CORDEAU.

ployed, these mains may be joined together as a measure of precaution. This is done in the manner shown in Fig. 8 by cutting the mains and putting caps on the ends and then uniting them again by means of a brass sleeve, the ends of which are firmly crimped to the detonating fuse. Around these brass sleeves the ends of the connecting line are wound in exactly the same manner as when attaching a branch to a main line.

Interesting details in regard to the use of this detonating fuse in France and Belgium may be obtained from the following articles, of which, for convenience, the writer has made a brief digest, viz.:

Commission du Grisou: Note sur l'amélioration de la sécurité dans les mines grisouteuses par emploi d'un nouveau dispositif d'amorçage des explosifs. *Annales des Mines*, 10th ser., vol. xii, p. 169 (1907). An investigation to determine whether the use of detonating fuse would increase safety in gaseous mines. Describes numerous tests and experiments and concludes with the following statement of the advantages of the detonating fuse:

1. No misfires, partial explosions, or burning of powder.
2. Allows reduction of powder charge. Estimated at 20 per cent.
3. Permits the use of explosives of very low sensitiveness.

Nouvelle Application du Cordeau Détonant aux travaux publics, by L. Barthélemy. *Bulletin de la Société des Ingénieurs civils de France*,

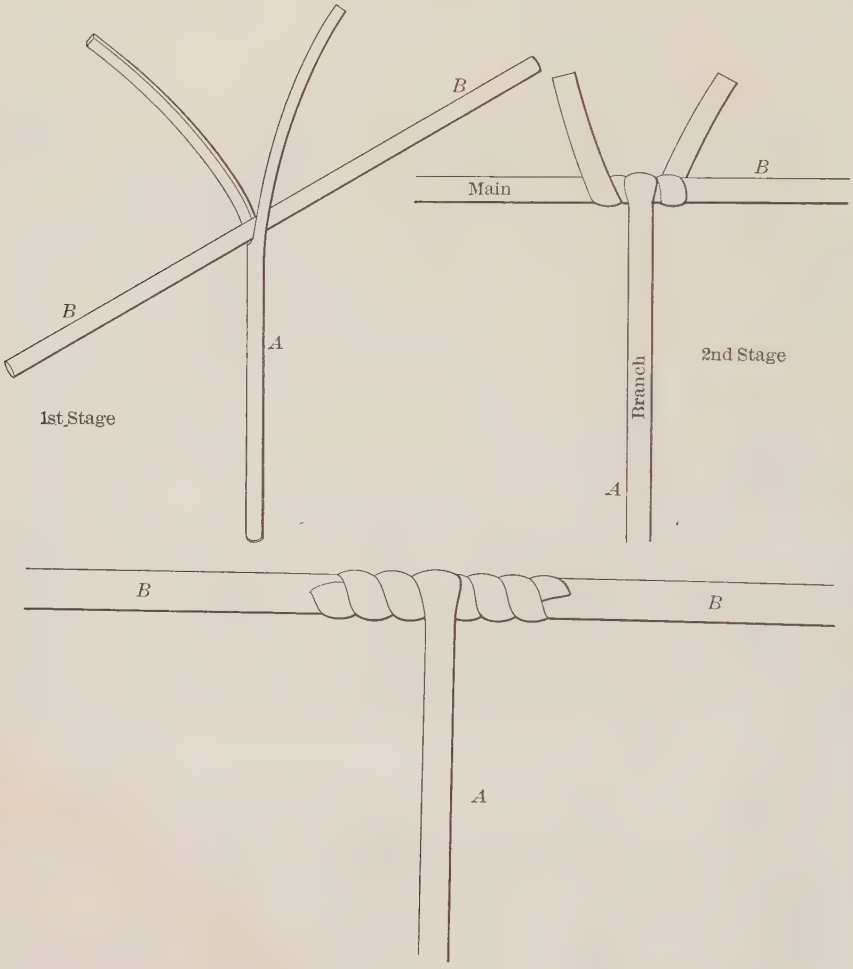


Fig. 6.—Method of Connecting Main Line and Branches.

November, 1910, p. 492. Describes the successful application of detonating fuse in the destruction of a 318.6-m. masonry tunnel on the Ypres canal in Belgium.

Shot 2,125 holes simultaneously. The charge was 3,250 kg. of melinite. The delicate part of the operation was to find a way to explode all of the holes at once. The first use of detonating fuse on a large scale.

Note sur le Cordeau détonant, by M. Fougerolles, Chief Engineer. *Comptes rendus mensuel de la Société de l'Industrie minérale*, February, 1908, p. 58. Description of tests made by Cia. des mines de Lens with permissible explosives and detonating fuse.

Conclusions: Compressed powders practical with easily observed precautions.

1. Be sure the hole is of large enough diameter to permit the charge to be forced tightly to the bottom.

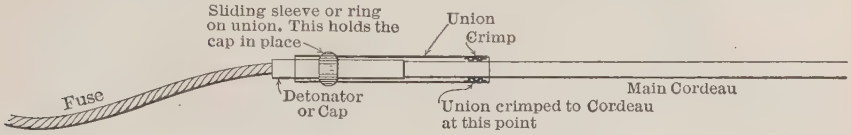


FIG. 7.—METHOD OF ATTACHING DETONATOR TO CORDEAU.

2. Make a fresh clean cut at end of the detonating fuse, and place it in contact with the exploder so that it cannot slip away while loading the hole.

No unexploded powder was found and it is claimed 20 per cent. of the explosives by weight and 10 per cent. of expenses were saved. Detonating fuse is well liked by the miners.

L'Explosion du Rocher de Formery. *L'Illustration*, May 31, 1913. Illustrated description of the removal of a large mass of rock, overhang-

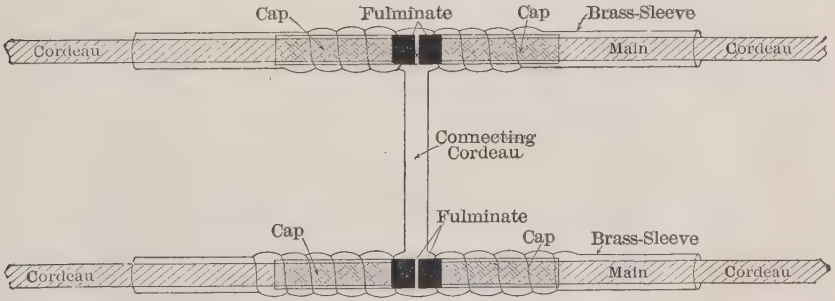


FIG. 8. —CONNECTING CORDEAU IS SPLIT AND WOUND AROUND THE BRASS SLEEVES IN THE SAME MANNER AS WHEN MAKING ORDINARY JOINTS.

ing the village of Formery in France, by blasting at one time 237 holes loaded with 92 per cent. nitroglycerin dynamite and connected with the detonating fuse. The whole was exploded with one detonating cap.

DISCUSSION

CLARENCE HALL, Pittsburgh, Pa. (communication to the Secretary*).—The introduction of Cordeau detonant to replace electric detonators used in blasting operations in this country has not met with the same

* Received Nov. 19, 1914.

success as it has in Europe. This has been due to the fact that the explosives manufactured and sold here are far more sensitive to detonation than the explosives used abroad. In Europe, and particularly in France, the explosives used in blasting operations are of the ammonia nitrate class and can only be detonated by the use of Cordeau detonant or a powerful detonator. Practically all explosives used in mining operations in this country are of the nitroglycerin class and when they are in good condition, and properly used, incomplete detonation rarely occurs.

In quarry operations, where the large drill hole method is employed and the charge of the explosive sometimes extends a distance of 30 ft., it has been found that one electric detonator will not always produce complete detonation of the entire charge in the drill hole; but when three or more electric detonators are employed throughout the charge, misfires or incomplete detonation rarely occurs. Cordeau detonant would no doubt be applicable in work of this kind, but in other blasting operations, where small charges of explosives are used and when sufficient stemming is employed to confine the charge, it is not believed that it could be advantageously or economically used. The cost of this detonating fuse is about 10c. to 15c. per yard, and it is necessary to use an amount that will extend from the bottom of the drill hole up to and beyond the mouth. It is true that this additional cost is a small part of the total cost of blasting in quarry operations, and the fact that this fuse will always insure complete detonation warrants its use in work of this kind.

The rate of detonation of Cordeau detonant is about 4,900 in. per second, but it cannot be assumed that greater energy will be developed when used with dynamites having high rates of detonation. However, when detonating fuse is used with dynamites of low rates of detonation, a greater shattering effect may be produced on the surrounding rock. When dynamites are used having rates of detonation higher than this fuse, then the only advantage in its use will be to insure complete detonation. In regard to increased efficiency resulting from the use of Cordeau detonant, this may obtain when an explosive of low rate of detonation is used or when explosives are used which have aged and are not in good condition. The statement that 20 per cent. reduction in powder charge could be made when the detonating fuse is used is questionable. The tests that the writer has made with Cordeau detonant do not show that greater efficiency obtains when used with small charges of explosives, such as the permissible explosives used in coal mining, or with other explosives in operations where shallow holes are employed.

S. P. Howell,* Pittsburgh, Pa. (communication to the Secretary†).—In discussing Mr. Souder's excellent paper on Cordeau-Bickford, a new

* Explosives Engineer, U. S. Bureau of Mines.

† Received Dec. 5, 1914. Published by permission of the Director of the U. S. Bureau of Mines.

safety detonating fuse, I shall have much favorable comment to make, for I believe that because of the safety features of this kind of fuse there is a field for its use in quarrying and in open-pit mining operations.

Among the advantages resulting from the use of this fuse, I desire to discuss particularly the following:

1. The elimination of the detonator or the electric detonator from the charged hole.
2. The reduction of "blind" misfires.
3. The increase in the disruptive effect of low-grade high explosives
4. The possibility of using relatively insensitive explosives.

The advantages mentioned are discussed in order below:

1. Of the three materials, trinitrotoluene (the explosive in the detonating fuse), 40 per cent. "straight" nitroglycerin dynamite, and mercury fulminate composition (commonly used in detonators and electric detonators), the least sensitive to impact and friction is trinitrotoluene; while the most sensitive is mercury fulminate composition. Bureau of Mines tests with the pendulum friction machine[2] show that a sample of trinitrotoluene is not exploded with a carborundum shoe operating over a steel anvil. The sensitiveness of the mercury fulminate composition is well illustrated by the many instances of accidents caused by pricking with a pin or needle the composition in a detonator. Consequently, the substitution of the detonating fuse for the detonator or electric detonator in drill holes is clearly a move toward safety, as the latter devices, because of their sensitiveness, have been held responsible for the many drill-hole accidents which would not have occurred had the holes contained nothing more sensitive than the ordinary high explosive.

2. In open-pit mining or excavating, and in quarrying, though misfires seldom occur, the possibility of their happening is always imminent. The "blind" misfire—one that is not known to have occurred—is clearly the kind most likely to cause an accident, the workmen being ignorant of the presence of the unexploded explosive, and, perhaps, of the detonator or electric detonator. The presence of an effective detonating agent—the detonating fuse—throughout the entire length of a charge in a drill hole must reduce, or perhaps eliminate, these "blind" misfires.

3. The disruptive effect[3] of high explosives depends in a measure upon the rate of detonation of the explosive. Tests at the Bureau of Mines show that the rate of detonation of a low-grade high explosive is increased to the rate of detonation of the detonating fuse extending through it. The following table of rates of detonation of certain samples

[2] See Hall, Clarence, and Howell, Spencer P.: Tests of Permissible Explosives. *Bulletin No.* 66, *U. S. Bureau of Mines*, Plate I, and pp. 15 and 16 (1913).

[3] See Hall, Clarence, and Howell, Spencer P.: The Selection of Explosives Used in Engineering and Mining Operations. *Bulletin No.* 48, *U. S. Bureau of Mines*, pp. 41 to 44 (1913).

of high explosives and of detonating fuse shows which particular high explosives may have their disruptive effect increased by the use of the detonating fuse.

Rates of Detonation of Trinitrotoluene Detonating Fuse and of Certain Samples of High Explosives

Class and Grade of Explosive	Diameter of Explosive Tested		Rate of Detonation	Authority
	Inches	Millimeters	Feet per Second	
Trinitrotoluene detonating fuse	19/64	4	13,750	Bureau of Mines.
Trinitrotoluene detonating fuse	15/64	6	15,170	Bureau of Mines.
Trinitrotoluene detonating fuse	....	.....	17,000	Manufacturer.
60 per cent. "straight" nitroglycerin dynamite	1¼	32	20,490	Bureau of Mines.
50 per cent. "straight" nitroglycerin dynamite	1¼	32	18,850	Bureau of Mines.
60 per cent. strength low-freezing dynamite	1¼	32	16,640	Bureau of Mines.
40 per cent. strength gelatin dynamite	1¼	32	16,210	Bureau of Mines.
40 per cent. "straight" nitroglycerin dynamite	1¼	32	15,650	Bureau of Mines.
30 per cent. "straight" nitroglycerin dynamite	1¼	32	14,920	Bureau of Mines.
40 per cent. strength ammonia dynamite	1¼	32	10,350	Bureau of Mines.
5 per cent. granulated nitroglycerin powder	1½	38	3,340	Bureau of Mines.

The data for Bureau of Mines rates of detonation for the explosives are taken from *Bulletin No.* 48, *Bureau of Mines*, p. 44.

4. As the percentage of nitroglycerin in low-grade dynamites is decreased, their sensitiveness to detonation is decreased, and hence when the nitroglycerin content has been reduced to a certain point their use becomes commercially impracticable. However, it would appear that such explosives, though having a low nitroglycerin content, could be used if the trinitrotoluene detonating fuse were used to detonate them. This would be likewise true of very insensitive explosives which contained no nitroglycerin.

Now, because this detonating fuse, as used in mining operations, is new in the United States, it seems highly desirable to emphasize such of the characteristics of the fuse as are most likely to cause failure when the fuse is improperly used, for this material, in common with other new materials used in mining operations, is likely to be misused.

Mr. Souder has emphasized (see p. 744) the importance of not joining the main fuse to the branch fuse; of joining the branch to the main at right angles; and of using the crotch method of attaching the branch to the main. Investigation has shown that the detonation of a branch will not detonate a main with certainty, and this confirms results obtained in Bureau of Mines tests[4] in which attempts were made to detonate the fuse through its lead covering with No. 6 and No. 8 electric detonators, but without success.

[4] See Hall, Clarence, and Howell, Spencer P.; Investigations of Detonators and Electric Detonators. *Bulletin No.* 59, *U. S. Bureau of Mines*, pp. 22 to 24, (1913).

The Bureau of Mines has tried various methods of attaching the branch to the main and has not been able to improve on the crotch method, while the simple method of attaching the branch alongside and parallel with the main resulted in failure.

By reason of the lead covering that incloses the trinitrotoluene of the fuse the material is rather heavy. For instance, a sample of 6-mm. fuse weighed 45 g. per foot, or approximately 10 lb. per 100 ft., and a sample of 4.3-mm. fuse weighed 27 g. per foot, or approximately 6 lb. per 100 ft. A sample of the same 6-mm. fuse 12 in. in length did not support a weight of 34 lb., and a sample of the same 4.3-mm. fuse 12 in. in length did not support a weight of 21 lb. It, therefore, appears that when the fuse is lowered into the deeper holes (50 ft. or more) care must be exercised to prevent jerking it, lest its own weight cause it to separate.

For use in these deeper holes the manufacturers have produced a covered fuse. The cover consists of closely wound cotton thread and adds greatly to the tensile strength of the fuse, so that lengths much longer than are feasible with the uncovered fuse can be used without danger of breaking. Except for the covering, the two types are identical.

Care should be exercised in tamping a hole containing the fuse and in choosing the stemming, for roughness in tamping may break the fuse, especially if the stemming be of hard and angular material.

HARRISON SOUDER (communication to the Secretary*).—The Institute is fortunate in persuading Messrs. Hall and Howell to discuss the use of detonating fuse, since their work with the U. S. Bureau of Mines gives an authoritative character to their opinion in regard to explosives.

Notwithstanding Mr. Hall's statement to the contrary, as a matter of fact in average daily American practice, misfires and incomplete detonations are quite common in both large and small drill-hole blasting and are the cause of many accidents.

A misfire, especially in the case of large well drill holes, is a very expensive proposition and adds a large element of danger to quarry work, where the steam shovel is used in loading the broken rock.

We know from our investigations that our own blasting troubles at Cornwall were typical of blasting troubles in many large and well-managed limestone and cement-rock quarries, with this exception: that our troubles were increased by the greater possibilities of the electric current being short circuited by the magnetic iron ore.

It is particularly for the large or well drill-hole blasting operations that we recommend Cordeau detonant. For our small drill-hole blasting we still prefer the electric detonator. While the use of Cordeau may not increase the energy of dynamite of equal or higher explosive rate, yet as a matter of fact it does increase the energy and efficiency of the total mass

* Received Mar. 7, 1915.

of dynamite in the hole by bringing about an instantaneous explosion of the whole charge, the effect being the same as if we had placed a continuous line or string of detonating caps from one end of the charge to the other.

The increased efficiency and decreased cost of blasting is made possible in that, with Cordeau detonant, a lower-strength and less sensitive explosive can be used to do the same work as an equal amount of higher grade, exploded in the old way; or the amount of high-grade explosive necessary under former methods can be reduced by using Cordeau in place of electric detonators, amounting in our Cornwall practice to about 10 per cent. saving. The saving of 20 per cent. is claimed by M. Fongarelles of the Lens Mining Co., operating French coal mines, and does not obtain here.

Furthermore, the time required to load a large blast is nearly cut in two by using Cordeau, and this is an item of great importance as it reduces expense as well as the hazard.

Mr. Howell in his discussion of the paper brings out the main features that should be emphasized, viz.: the first and most important gain from the use of Cordeau is safety, and incidentally there is at the same time a saving in expense.

It not only eliminates the electric detonator from the charge and places it outside the drill hole, where it is under observation and can be removed in case of trouble, but it does away with electric lead wires around the mine or quarry, where live current is used in blasting. In some recent instances special power equipment was installed to provide current for this blasting. This expense is eliminated.

The use of old blasting batteries, which are generally overrated and overloaded and are a fruitful cause of misfires, is done away with. Blind misfires are almost impossible in open bench work. Under the old method the presence of wires from the detonator might mean that the detonating cap had exploded properly or it might mean that the detonator was still buried in a stick of dynamite under the rock, waiting to be struck by a pick or dipper tooth, whereas the presence of a piece of Cordeau indicates at once a misfire.

The rates of detonation given by Mr. Howell indicate that the rate for Cordeau is about equal to that of 30 and 40 per cent. nitroglycerin dynamite, which is the grade of dynamite in common use in quarries. The French experimenters claim a higher rate, as do also the manufacturers.

As to the methods of using Cordeau, we are free to confess that we have had some troubles in using it. The greatest trouble came from the inferior quality of the Cordeau which was imported from abroad. The trinitrotoluene compound, or T. N. T. as it has been abbreviated, was not stable and it was found after a while that it turned from pure white to

yellow brown. This condition could be disclosed by a fresh cross cutting or by peeling some of the lead casing from the inclosed explosive.

The manufacture of this material is now carried on in this country, since which time trouble from this cause has ceased. Nevertheless, each lot is inspected before being used.

Another trouble was due to the breaking of the Cordeau in loading, especially in deep holes. The American makers immediately made it with a cotton sizing or covering over the lead, greatly reducing liability to break. Occasionally, some defective lead covering is discovered. This renders the fuse liable to break easily and also permits moisture to enter.

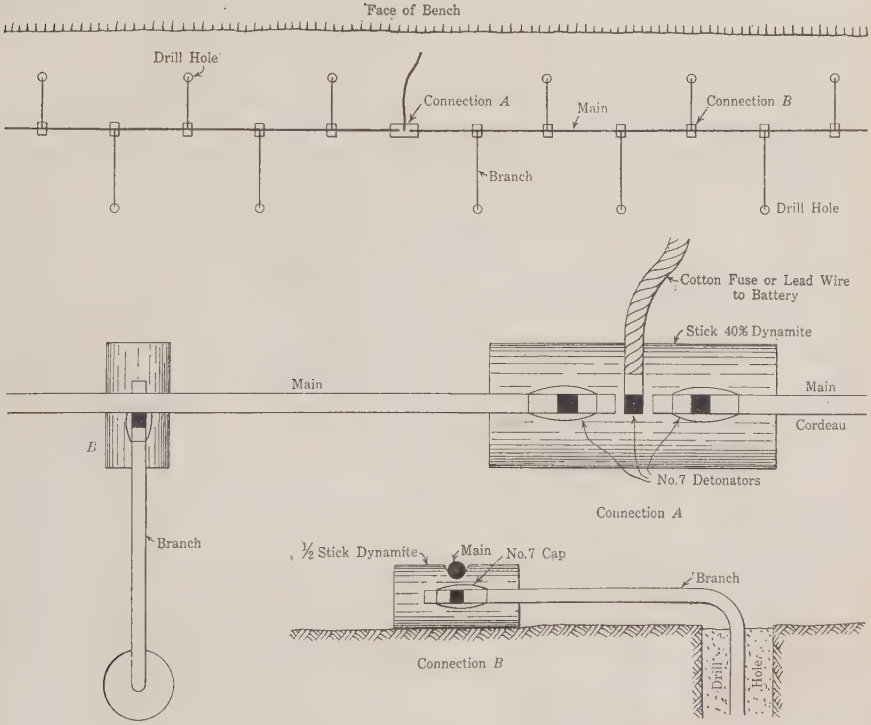


FIG. 9.—PRESENT METHOD OF USING CORDEAU.

The original Cordeau used was 6 mm. in diameter and an effort to reduce the cost was made by reducing the size to 5 mm. This was found unsatisfactory and about 5.5 mm. is now generally recommended.

Our feeling is that for big shots 6 mm. diameter is best, especially if the crotch method of uniting branch and trunk lines is used.

Some trouble was encountered in making the crotch connection of branch line with main leader. The extreme care necessary for this operation, especially with the smaller diameter Cordeau, was not always exercised. Also it consumed much time. After many experiments on a

small scale and actual trial in large blasts, the method given below was evolved and has proved on the whole most satisfactory for ordinary practice.

After the holes are loaded, allowing about 10 in. of Cordeau to project from each hole (in case of a single row), a brass ferrule is placed on the end of each fuse, and a No. 7 blasting cap is inserted in it to bear against the end of the fuse. Then a half stick of 1¾ in. diameter 40 per cent. dynamite is slipped over the end of each branch fuse, so as to inclose the No. 7 cap. Then the main lead Cordeau is strung along the holes and notched into each half stick of 40 per cent. dynamite at the end of each branch fuse; and the connection is mud capped or covered over with a little mound of clay or sand and patted down. This is carefully done for each hole. Then the two ends of the lead fuse are brought together and a No. 7 detonating cap attached to each end, and these are inserted in a stick of 40 per cent. dynamite. Then a piece of ordinary cotton fuse, say 2 ft. long, is tipped with a detonator and this inserted in the stick of dynamite, which is then mud capped. When ready the fuse is lighted in the ordinary way and sets off the whole charge. In case two rows of holes are shot at one time, as is our ordinary practice, the connections are made so as to insure simultaneous explosion of all holes as nearly as possible, since we found that the short interval of time between the explosion of the back row and the front row, in cases where the back row went first, permitted rock strata to move sufficiently to cut off the front row of fuses in some cases. Where many rows are blasted at once many cross connections should be made between the main trunk leads.

Where a limited number of holes have been shot we have modified the practice, attaching to the Cordeau from each hole an electric exploder, wiring the holes in series, and shooting with a battery in the old-fashioned way. The sketches, Fig. 9, show the method we now use.

In conclusion, we may state that in our experience and judgment the introduction and use of Cordeau detonant marks a great advance over former practice, in making for safety and economy in blasting operations, especially for well-drill or churn-drill deep-hole blasting.

The use of Cordeau detonant has now become quite customary in the large cement-rock quarries of eastern Pennsylvania, and we believe further investigations of this material by the U. S. Bureau of Mines would be of much service and value to those operating open-cut mines and quarries.

The Safety Movement in the Lake Superior Iron Region

BY EDWIN HIGGINS, PITTSBURGH, PA.

(Pittsburgh Meeting, October, 1914)

Introduction

It is the purpose of this paper to set forth the relation and functions of the various organizations and institutions engaged in the promotion of safety in the iron mines of the Lake Superior region; also to indicate the value of this work. Ten to 15 years ago there was practically no organized safety work; the accident rate was high, excessively so in some districts, and dangerous practices and conditions existed in many of the mines.

During recent years, however, a gradual change for the better has been made. The chief causes of this change have been (*a*) public opinion, which has set the stamp of disapproval on the disregard for human life; (*b*) certain State laws which have made the operators responsible in dollars and cents for injuries to workmen; and (*c*) the humanitarian attitude of many of the operators, who have always decried the great loss of life in the mines.

To-day the Lake Superior region as a whole stands second to no other metal-mining district in the United States in its efforts to promote the welfare and safety of the miner. Dangerous practices in and about the mines are fast disappearing. The operators are ready and eager to adopt any expedient, rule, or device that holds forth a reasonable promise of reducing the hazards of the miner. To-day the value of a mine captain or shift boss is reckoned, not alone on his ability to "get the ore," but also on his capacity for reducing accidents. While the progress made has been remarkable, there still remains much to be done, for the accident records of the Lake Superior iron mines, while lower than those of the chief metal-mining regions of the United States, still compare unfavorably with those of the metal mines of practically all foreign countries.

Organizations Engaged in Safety Work

There are five organizations, or institutions, engaged in safety work in the Lake Superior region, viz.: The mining companies; county mine

inspectors; co-operative range committees; Lake Superior Mining Institute; Federal Bureau of Mines.

While each of these bodies covers well-defined features of the work, their activities are correlated to a certain extent. The mining companies, primarily, are concerned with conditions in their respective mines, but they co-operate with and receive aid from the other agencies mentioned. The county mine inspectors, whose duty it is to see that the mines are operated with due regard to the State laws, are of great assistance to the mining companies in suggesting remedies for dangerous conditions or practices. The co-operative range committees, made up as they are of officials and employees of all companies operating within a given radius, are a benefit to the body of operators of their respective ranges. The committee of the Lake Superior Mining Institute on practices for the prevention of accidents, concerns itself chiefly with questions of safety of widespread interest; it of course has the co-operation of the other bodies interested. The Federal Bureau of Mines is chiefly active in training miners in the use and care of rescue apparatus and in first aid to the injured; also, it has conducted special investigations of certain problems having to do with safety and efficiency in the mines.

The unity of ultimate purpose and the strong co-operative spirit which have characterized the work of these five bodies have been important factors in the development of the conditions of to-day.

The Work of the Mining Companies

The bulk of the cost of the safety work has been and is still borne by the mining companies. In the main, their work has been to provide protective devices in and about the mines, and to educate the miners, by means of rules and regulations and various other methods, so as to enable them to protect themselves from injury. These objects seem easy of accomplishment, but there are obstacles of various kinds continually arising to hinder progress toward the desired end. Some of the most serious factors, most of which still exist to a greater or less extent, are the prejudice of the old-time miner or boss toward safety regulations; carelessness and lack of interest in safety work on the part of the miners, and even the bosses; and at times the scarcity of labor, which necessitates the employment of less skilled and oftentimes ignorant men.

The greatest problem to-day is not to secure knowledge of how safety work should be conducted, and what protective devices to use, but how to get the miner to use these methods and devices. Probably not 10 per cent. of the miners, if subjected to an examination, would show even a passing knowledge of the contents of the books of rules and regulations. Safety devices, provided at great expense, are often found removed from their places, or disregarded entirely.

The first problem of the mining companies was to provide for an organization to carry on the safety work; then to devise means of protecting the miner and of educating him and securing his co-operation in the prevention of accidents.

Organization

Practically every mining company in the district now has some individual, or organization, whose duties relate solely to safety work. The larger companies have well organized safety departments. The following brief outline of safety organizations will indicate the usual procedure in this connection:

1. The mines covered by this organization are large and are all situated on one of the iron ranges of Michigan. The safety department is in charge of an inspector and it is his duty to inspect all mines as frequently as possible and submit reports and recommendations to the manager. Periodical trips are made in and about the mines by a committee of mine foremen consisting of three members, each of whom is selected from a different mine. The inspector accompanies this committee and incorporates its recommendations in a report. Another committee, having similar duties, is made up of workmen. The activities of this committee, however, are limited to the mine from which it is selected. The members are changed after each inspection, so that in time all employees are given a chance to criticize conditions in and about the mine.

All accident reports and safety recommendations are considered by a committee of mine superintendents, the head mining captain, master mechanic, assistant auditor, secretary of the pension department, safety inspector, and the manager, who acts as an *ex-officio* member. This committee meets once a month and confirms or rejects safety recommendations.

In addition to the above committees, there is one more made up of three mine superintendents. This committee investigates all fatal accidents and makes a report thereon to the manager.

2. The following form of organization is employed by a company operating both large and small mines at various scattered points. The department is under an inspector, who, with the assistance of three experienced miners, inspects each mine of the company at least twice a week. After a mine is examined, a report, including any recommendations thought necessary, is sent to the safety inspector. The safety inspector in turn makes a weekly report to the superintendent, who looks after all recommendations having to do with upkeep. The safety inspector makes a monthly report in triplicate to the manager in which recommendations are submitted for approval. Such recommendations are made out to the head of the department concerned. When approved by the manager, one copy is returned to the safety inspector, to be kept by him

until the indicated improvement is made. Two copies go to the superintendent, who keeps one and sends the other to the head of the department concerned. On the completion of the improvement, the head of the department sends the recommendation back to the superintendent, who then destroys his copy and sends the indorsement of completion to the safety inspector. The safety inspector destroys his record and files in its place the report showing that the improvement has been made. This report is in the form of a printed card with blank spaces filled in according to the needs.

All company bosses and first-aid men meet once every two months to discuss all accidents that have occurred during the previous two months. The subjects discussed at these meetings are safety, sanitation, first aid, and welfare.

3. A similar organization to that described above is maintained by a company operating small groups of mines at scattered points. A chief inspector is in charge of the safety work at all the mines. The foremen's safety committee, consisting of four foremen from the mines of a certain district, works directly under the chief inspector. This committee makes a trip every three months through all mines of the district. Its personnel is changed after each inspection trip. The committee reports to the inspector, who, in turn, includes this report in his recommendations to the superintendent; a copy of the report also goes to the general manager.

4. This organization operates in connection with one large mine. Inasmuch as the organization was changed after the safety work was well under way, it may be well to point out the various steps in perfecting the organization. An engineer was placed in charge of a department of efficiency and safety. He first made a thorough study of conditions in the mine and determined the principal causes of injury to the men employed. Finding that the greatest number of accidents occurred from falls of rock and ore, and from men falling down unprotected places in the mine, timber inspection was doubled and every place in the mine where it was thought that there was a possible chance for a fall to occur was timbered. All open places were protected by means of doors or gates. This movement effected an immediate and marked falling off in the number of accidents from the causes mentioned.

Finally, three assistants were added to the department and each one of them was given a certain feature of the work to look after. This was necessary on account of the magnitude of the operation. The safety and efficiency work then developed into daily inspection trips by all the members of the department. Reports were made by them to the head of the department, who considered recommendations made and obtained immediate action thereon through consultation with the manager. Lately these daily inspection trips have been abandoned; the members of the organization now average two or three trips a week through the mine.

Daily meetings, attended by the manager, superintendent, head of the efficiency and safety department, and mine captains, are held. Here all matters pertaining to efficiency and safety are discussed. As these meetings are held in the morning, it is possible to hear the reports of the shift bosses to the mine captain. In this way daily happenings and conditions in the mine receive prompt attention.

Protective Methods and Devices

Under the head of protective methods and devices it may be said, in general, that the work proceeds along the following lines:

A study of mining and timbering methods with a view to greater safety.

A study of safe methods in every department of mine work.

The protection of dangerous places in the mine.

Protective coverings for all exposed parts of machinery.

Installation of safety devices in and about the mine.

The provision of such tools and appliances as will result in the maximum safety to the employees.

Installation of devices for protection against fire.

The inspection of all working places, shafts, and machinery at stated intervals.

The method of carrying on the inspection work is indicated in the descriptions of the various types of organizations. Nearly all of the protective methods and devices are suggested through information gained on the inspection trips.

Welfare and Educational Work

It has long been recognized that various measures looking to the welfare of workmen are most desirable from many standpoints. In the study of safety work it has developed that any line of work that will serve to secure the co-operation and confidence of employees is of the greatest value in promoting safety. Without the co-operation of the workmen safety work cannot advance, and in order to gain this co-operation it is necessary first to secure the confidence of the men. Indeed, many students of safety believe that the winning of this confidence and co-operation constitutes almost the entire safety problem. Lake Superior mining companies are spending as much money in welfare and educational work as on the actual installation of safety devices, if not more.

Among the activities of several of the large companies looking to the welfare of the miner may be mentioned the following:

The provision of a pension fund for workmen who have grown old in the service of the company; erection of club houses where the workmen may spend their idle hours; the building of model towns and houses for

the use of workmen; the offering of cash prizes for the best-kept premises, lawns, and flower gardens; the building of expensive and commodious dry or change houses. Practically every iron mine in the Lake Superior region is provided with a dry or change house; some of these are models of cleanliness and perfection in other details.

Among the more important educational features designed to secure the co-operation of the miner, may be mentioned the following:

Rules and Regulations.—Practically all of the large companies issue to their workmen books of rules and regulations. In some cases these books are printed in from eight to ten different languages. In general, they cover all departments of mine work. These rule books in the past have contained many regulations that were not enforced, but of recent years there has been a noticeable tightening in this respect and the tendency now is to eliminate many of the useless rules and to be more strict in the enforcement of those retained. There is still room for improvement, however, in the matter of enforcing rules.

Cash Bonuses.—The matter of cash bonuses to bosses and others for the prevention of accidents has not, in the past, been given serious consideration. During the last year, however, the managements of several companies have come to the belief that an equitable system of cash bonuses will be of value in reducing accidents. There is only one company that has really put the system into operation. While the method used by this company cannot be termed strictly a bonus system, it is in principle the same thing. By paying to shift bosses salaries that were larger than those paid by other operators, but insisting that these shift bosses be safety enthusiasts, this company has conducted its operations with a minimum percentage of accidents for the district. One large company is preparing to adopt a system of paying large cash prizes to the shift bosses turning in the best records as to the number of men killed or injured while working under their supervision.

Publicity of Accident Records.—A method of attracting the attention of the workmen to the hazards of their employment, which has been adopted to a certain extent, is to post placards at various points in and about the mine calling attention to all serious accidents that happen, and pointing out how they might have been prevented. In some cases sketches and photographs are used as illustrations with these placards. At some mines the records of the different shift bosses, as to the number of accidents that happen to the men working under them, are posted in conspicuous places.

Pay for Safety Suggestions.—It is the general practice in the district to offer cash rewards for suggestions from workmen that may lead to safer working conditions.

First-Aid and Rescue Instruction.—Through the activities of the mining companies, in co-operation with the Bureau of Mines, miners have been

trained in first-aid and rescue work at practically every mine in the district. This work has not been limited to three or four men at a mine, but in many cases has been continued so that in some cases as many as 20 per cent. of the employees have received first-aid training. Training in rescue work, however, has usually been limited to from five to ten men at a mine, or group of small mines.

Work of the County Mine Inspectors

The various counties of the Lake Superior district in which mines are situated, are each provided with a mine inspector. In some cases this inspector is allowed to employ such assistants as may be necessary in his work. The various counties require that every mine shall be inspected at stated periods. The principal duty of the county mine inspector is to see that the mines are operated in accordance with the laws of the State. He is empowered with authority to cause the closing down of a mine in case the management refuses to comply with his demands. In reality, these inspectors have rendered services in excess of what is demanded of them by law. Through the study of conditions in the mines they gain information that is of common value; such information is disseminated to mine operators in the form of suggestions for the promotion of safety. One inspector has gone so far as to make and keep records of safety devices and methods. These he has communicated to all the operators in his county through the medium of circular letters and blueprints. The efforts of these county mine inspectors and their assistants have gone far toward promoting safety in the mines of the Lake Superior region.

Work of the Co-operative Range Committees

There have been organized on the various iron ranges what may be termed co-operative range committees. During the year 1913, five such organizations were perfected. It is believed, however, that only three of them are now holding regular meetings and doing efficient work. These committees are made up of mine superintendents, mine captains, shift bosses, safety inspectors, men in charge of first-aid and rescue work and other mine employees interested in safety work. The committees have the backing of the management of the mines, which stand the bulk of the expense of carrying on their work. The organizations are made up of representatives of practically all companies within a given radius. For instance, the Gogebic Range Mining Association includes in its membership representatives from all companies operating on the Gogebic range. The purposes of this organization, as set forth in its by-laws, are to promote social intercourse and the interchange of ideas on all subjects of mining interest, for the mutual benefit of its members; and to perpetuate

efficiency, welfare, safety, mine-rescue work and first aid to the injured in and about the mines. These purposes are accomplished by (*a*) social meetings; (*b*) remarks, discussions, and the presentation of papers by members of the organization at different times; (*c*) occasional visits to the different mines, plants and properties upon invitation of the management of same; and (*d*) occasional competitive meets for crews trained in mine-rescue work and first aid to the injured.

The greatest good done by these committees has been in disseminating, for the benefit of all the mining companies, information relating to safety, efficiency and other mine work. They have developed in some cases into organizations representing the needs and wishes of entire communities. Their opportunity for the promotion of the general welfare is unlimited.

Work of the Lake Superior Mining Institute

The Lake Superior Mining Institute some years ago established a committee on practices for the prevention of accidents. This committee holds special and stated meetings at which are considered important problems relating to safety and mine operation. Among important subjects upon which recommendations have been made may be mentioned the uniformity of mine-accident reports. Under present conditions the mining companies make reports of accidents to several different organizations, all of which require different classifications of accidents. The work thus entailed is enormous. By providing a uniform type of report this undue work may be eliminated. Furthermore, reports of the county mine inspectors, although satisfactory as far as the needs of each county are concerned, are made in such form that it is impossible to make intelligent comparisons of records of the various counties. An effort will be made, through this committee, to standardize all reports.

In addition to other important considerations, a study is now being made of mine rules and regulations, with a view to eliminating unnecessary regulations and including others that appear to be of paramount importance.

This committee recently considered and made recommendations that led to the holding of a first-aid contest at Ishpeming, Mich., during August, 1914. This meet was attended by teams from all over the Lake Superior region.

Work of the Federal Bureau of Mines

The Federal Bureau of Mines has headquarters at Ironwood, Mich., established in November, 1912. It has recently acquired, through lease, a small tract of ground on the right of way of the Chicago & Northwestern Railway, and a spur has been built thereon for the accommodation of the rescue car. Arrangements have been practically perfected under which

the operators of the Gogebic range will erect a building containing the necessary office room and housing for the rescue car.

The Bureau's representatives in the district have comprised a district engineer, a foreman miner, and a first-aid miner. The rescue car has been active in training the miners of the entire region in first aid to the injured and rescue work. It has not, however, been continuously engaged in this work owing to lack of available funds. Up to the present time there have been trained in the entire district a total of approximately 700 men in first aid to the injured, and 400 men in the use of oxygen breathing apparatus. These men have trained others and it is estimated that there is now a total of 2,000 men in the district who have received training in first aid to the injured, and 1,000 men who have received training in the use of oxygen breathing apparatus.

In addition to this work of training, the district engineer has visited and examined a large proportion of the mines of the region. These examinations were followed in some cases, at the request of the management, by recommendations for increasing safety in the mines. Special investigations undertaken include those having to do with ventilation, mine fires, organization and conduct of safety work, the use of mine sign boards, and hoisting signals. Papers have been written on these subjects for publication by the Bureau and by various mining institutes. Other educational work carried on comprised illustrated lectures to the miners in and about the mines visited by the rescue car. The district engineer brought about the organization of the co-operative range committees mentioned on preceding pages.

The activities of the Bureau of Mines have doubtless been instrumental in furthering the work of safety. Its chief function has been in stimulating activity in safety, rescue, and first-aid work. The moral effect of the presence of the car and its attendants has served to call the attention of the miner to the fact that the subject of safety and first aid is of sufficient interest and importance to cause the government to take an active hand in the work. The Bureau of Mines' representatives have worked in co-operation with the operators and every one else engaged in safety work in the region, and have been favored with most courteous treatment and co-operation in all their efforts. It might be added here that the writer has, for the past two years, served as district engineer of the Bureau in the Lake Superior region.

Results of the Safety Movement

It is not possible at this time to prepare a statement that will indicate the full measure of benefit derived from the practice of safety work in the Lake Superior region. The safety movement is practically in its infancy, and a period of years must elapse before any fair estimate can be made of

the actual good resulting from it. That all classes of accidents are steadily decreasing is shown by tables submitted herewith; the next few years should show a proportionate or even greater decrease.

Inquiries as to the reduction of accidents through safety work directed to all the operators of the district were answered, in a majority of cases, by the statement that records are not yet available; there were many indefinite replies stating that accidents had decreased; no replies were received stating that accidents had increased. Personal inquiry by the writer brought forth statements from the most important operators indicating that safety work has caused a material reduction in accidents, and that they are all desirous of continuing the work. In many instances a great deal of enthusiasm was displayed. An interesting statement regarding fatal injuries sustained by employees was received from the management of a large iron mine. It may be remarked that this company began active safety work early in 1912. The statement appears in Table I.

TABLE I.—*Record of Fatal Accidents in an Iron Mine*

Year	Total Number of Men Employed per Day	Number of Fatal Accidents per Year	Fatal Accidents per Year per 1,000 Men Employed
1889	185	2	10.81
1892	224	2	8.93
1893	259	1	3.86
1898	312	11	35.26
1905	372	3	8.07
1906	719	16	22.25
1907	731	5	6.84
1908	874	3	3.43
1909	1,050	8	7.62
1910	1,317	12	9.11
1911	1,235	6	4.86
1912	729	5	6.86
1913	1,548	5	3.23

No accurate records exist of accidents in the Lake Superior mines for a period of years. The Bureau of Mines only began the collection of accident statistics in metal mines in 1911. However, a compilation showing the fatal accidents for ten years previous to 1911 has been made by O. C. Davidson, General Superintendent, Oliver Iron Mining Co., Iron Mountain, Mich. Mr. Davidson reviewed the county mine inspectors' reports from Sept. 30, 1901, to Sept. 30, 1911. With the data thus obtained, and tonnages based on statements of shipments published by the *Iron Trade Review*, he was able to prepare this interesting statement. The compilation is submitted herewith as Table II. Attention is directed to the low death rate indicated in Dickinson county and the high rate in Iron county.

TABLE II.—*Summary of Fatal Accidents in Michigan Iron Mines from Sept. 30, 1901, to Sept. 30, 1911*

County	Tons of Ore Mined 10 Years	Total Number of Employees 10 Years	Number of Fatal Accidents Reported	Tons of Ore Produced per Fatal Accident	Fatal Accidents per 1,000 Men Employed
Dickinson	22,601,474	31,836	84	269,065	2.638
Marquette	36,721,389	57,161	248	148,070	4.339
Gogebic	29,191,952	42,471	226	129,168	5.321
Iron	17,986,380	20,962	158	113,838	7.537
	106,501,195	152,430	716	148,745	4.697

The number of fatal accidents and other data for the years 1911 and 1912 are shown in Table III. It may be noted from this table that Minnesota stands first and Michigan second in the column showing the number of men killed per thousand employed. The other States included in this table comprise the chief metal-mining States of the country. The figures for Michigan and Minnesota serve the further purpose of showing the decrease in fatal accidents as compared with the 10 years indicated in Table II. These figures were obtained from *Bureau of Mines Technical Paper No.* 61, Metal Mine Accidents in the United States During the Calendar Year 1912, by A. H. Fay. Figures for 1913 are not yet available. From county mine inspectors' reports, however, it is probable that the accident rate for the Lake Superior region during 1913 will show a marked decrease.

In summing up the beneficial results of the safety movement, it may be said that by far the most valuable accomplishment has been the reduction in the number of deaths, and serious and permanent injuries. This, of course, has been brought about through the improved working conditions in the mines, more vigorous inspection, and the reduction of the hazards to the worker through the use of various safety devices. Bodily suffering has been reduced, the earning power and efficiency of the worker have been increased, and mental suffering and hardship on the part of widows, orphans and other dependents have been lessened.

Not only have beneficial results been forthcoming from a humanitarian standpoint, but also from a financial standpoint. When a miner is injured the money that he can contribute toward the support of his dependents is curtailed in proportion to the seriousness of his injury. If the miner is killed the support of his dependents devolves upon others, thus giving them a double burden to bear. The employer also sustains financial loss, both in hospital expenses and in the payment of compensa-

TABLE III.—*Fatal Accidents in Various Metal-Mining States During the Calendar Years* 1911 *and* 1912

State	Number of Operators Reporting		Total Number of Employees		Number Killed		Killed per 1,000 Employed	
	1911	1912	1911	1912	1911	1912	1911	1912
Alabama	25	20	4,101	4,827	10	33	2.44	6.84
Arizona	352	479	12,768	15,591	70	67	5.48	4.30
California	855	1,048	10,877	10,312	38	40	3.49	3.88
Colorado	660	624	10,404	8,892	43	48	4.13	5.40
Idaho	513	639	4,801	6,229	23	29	4.79	4.66
Michigan[a]	74	79	31,584	29,469	134	96	4.24	3.26
Minnesota[b]	40	43	16,548	16,559	76	50	4.59	3.02
Wisconsin[c]	8	11	1,157	2,338	2	9	1.73	3.85
Montana	332	405	13,346	13,340	62	50	4.65	3.75
Nevada	472	554	6,210	7,547	50	34	8.05	4.51
Utah	295	336	7,710	8,458	49	41	6.36	4.85

[a] In copper mines, 14,893 men employed, 44 killed, 2.95 killed per 1,000 employed. In iron mines, 14,378 men employed, 52 killed, 3.62 killed per 1,000 employed.

[b] All iron mines.

[c] All iron mines.

tion. On the Marquette range, when a miner is killed, it has long been a custom for the entire mine force to cease work until the victim of the accident has been buried. The loss caused in this way amounts to approximately $2.30 per day per man involved, and an average of $500 in fixed charges to the company for every fatal accident.

Another source of financial loss is that sustained by the tax payers for the maintenance of courts for the trial of damage suits. Investigation discloses that in one Minnesota county $75,000 per annum has been spent in this manner.

In addition to the reduction in the loss of life, and the saving in money to both the miner and the operator, other benefits have developed. There appears to be a better understanding between employer and employee and the miners are beginning to realize that the safety work is being done for their benefit. Throughout the district a strong spirit of co-operation is noticeable, not only between the miner and employer, but between the officials of the various mining companies and mining districts. Information regarding safety and efficiency work is exchanged between operators with the utmost freedom, and there is hardly a mine in the region that is not open for inspection as far as safety devices and methods are concerned.

Cost of Safety Work

It has been impossible to arrive at even an approximate estimate of the amount of money spent on safety work in the Lake Superior district. Inquiries directed to all of the companies by the Bureau of Mines in 1913 were in a large majority of cases answered by the statement that no record has been kept of the money spent. Of course, many of the companies that have separate safety departments have kept records, but even some of these are incomplete in that they do not include money spent before the completion of the safety organization. Accurate figures should be obtained in the future, however, as safety work is becoming more and more a definite part of mine operation.

Perhaps an extreme case in the matter of expenditures in safety work is reported in the following statement taken from *Bureau of Mines Technical Paper No.* 30, "Mine-Accident Prevention at Lake Superior Iron Mines, by Dwight E. Woodbridge:

Cost of Safety Devices Installed by One Lake Superior Iron-Mining Company in 1911.

Menominee range, Michigan and Wisconsin	$8,196
Gogebic range, Michigan and Wisconsin	15,733
Marquette range, Michigan	19,937
Vermillion range, Minnesota	11,327
Baraboo district, Wisconsin	222
Mesabi range, Minnesota:	
Hibbing district	18,770
Chisholm district	11,142
Adams district	8,001
Fayal district	8,525
Virginia district	14,815
Canisteo district	21,152
	$137,820

Table IV, showing expenditures per year of various companies for safety work, will serve to throw some light on this subject, although the figures are extremely variable. It may be noted that the figures in this table account for only 5,480 men, whereas there were employed in the iron mines of Michigan, Minnesota, and Wisconsin, during 1912, a total of 33,275 men. From this table it appears that the lowest sum spent per year, per man employed, was $1; while the highest is more than $48. The average for the whole is $10.10 per year per man employed, which is probably a high average for the region.

One company, employing over 2,000 men, spends approximately $6 per year per man employed. This company has a well-organized and efficient safety system. Another large company spends slightly more than this. In this connection it must be considered that in the beginning,

TABLE IV.—*Expenditures during* 1912 *for Safety Work of Some Lake Superior Iron-Mining Companies*

Annual Production, Long Tons	Number of Men Employed Daily	Amount Expended
237,358	297	$5,075
225,000	265	2,000
75,000	130	1,000
350,000	110	3,000
400,000	150	3,000
250,000	310	15,000
28,100	40	230
169,708	154	885
210,068	158	480
155,000	110	903
128,465	120	4,850
312,000	85	1,000
1,250,000	1,000	1,000
2,033,242	2,178	13,715
3,700	38	311
124,000	175	1,500
102,400	160	1,407
	5,480	$55,356

when safety devices are being installed, the expense is greater than in succeeding years. It is probable that a fair average for maintaining safety work, after the preliminary work has been done, would be in the neighborhood of $5 per year per man employed. During the year in which safety devices are installed, if an elaborate system be adopted, the cost may run as high as $10 or more per man.

DOES SAFETY WORK PAY?

Answering the question as to whether or not safety work pays, one may say without hesitation that it pays enormously from a humanitarian standpoint. There is no argument here. The records of many coal companies and other organizations which have practiced safety work for a number of years show undoubtedly that it pays also from a financial standpoint. As to figures on this phase of the question in the Lake Superior district, nothing accurate is possible of compilation. Without the cost of the work as a basis, acceptable figures cannot be submitted.

However, there is a method of arriving at approximate figures, based upon actual conditions in the Lake Superior region. Before going into this, it seems proper to submit some figures regarding the compensation

that must be paid to miners for various classes of injuries. The States of Michigan, Minnesota, and Wisconsin all have in force a workman's compensation act. The following information is from the Public Acts of the State of Michigan, the provisions of which differ little from those of Wisconsin and Minnesota.

In case of a fatal injury to an employee, the employer must pay to the dependents of the injured a weekly sum equal to one-half his average weekly wages, but not more than $10 or less than $4 a week, for a period of 300 weeks from the date of the injury.

In case of permanent disability resulting from injury, the employer shall pay to the injured a weekly compensation equal to one-half his average weekly wages, but not more than $10 or less than $4; in no case shall the period covered by such compensation be greater than 500 weeks, nor shall the total amount of such compensation exceed $4,000.

Thus it may be seen that a permanent injury may cost the company more than a fatal injury. For injuries resulting in temporary disability, the injured receives a weekly compensation equal to one-half the difference between his average weekly wages before the injury and the average weekly wages which he is able to earn thereafter, but not more than $10 a week; and in no case shall the period covered by such compensation be greater than 300 weeks from the date of injury.

The above provisions are followed by a long list of payments to be made to the injured in case of the loss of a finger, a hand, a foot, an eye, etc.

In order to show what saving may be effected through the practice of safety work, let us assume an iron mine employing 1,000 men per day, in which no money is spent for safety work. Past experience has pointed out that a mine of this size, making no attempt to prevent accidents, may easily make the following yearly accident record:

Number of men killed	6
Number of men seriously injured	40
Number of men slightly injured	250

A serious injury may be considered as one that incapacitates the workman for more than 20 days; a slight injury one that incapacitates him for less than 20 days. The total cost in compensation for this sum of accidents, based upon the compensation stated above, may be as follows:

6 men killed, at $2,500 each	$15,000
2 men permanently injured, at $3,000 each	6,000
13 men, average disability 20 weeks, at $7 per week	1,820
25 men, average disability 8 weeks, at $7 per week	1,400
250 men slightly injured, average disability 1 week, at $7 per week	1,750
Legal fees, hospital and other casualty expenses	15,000
Total	$40,970

This tabulation does not include payments for the loss of hands, feet, etc. In total amount it is below many records that have been noted by the writer.

Now, let us suppose that this company had practiced safety work, and that the death rate was three men killed per 1,000 employed (approximately the Minnesota 1912 rate) and the injuries proportionately lower. This would mean a reduction of one-half, or a saving of approximately $20,000.

Of course the safety work will cost something, but even if it amounted to $10 per year per man employed, or $10,000 in this case, there would still be a balance of $10,000 saved.

Conclusions

It has not been the intention in this paper to convey the idea that the Lake Superior iron ranges comprise the only metal-mining region in the United States where efficient safety work is done. It is acknowledged that there are individual mines elsewhere that can show accident records just as good, and possibly better than those of some of the Lake Superior mines; also that the safety movement is gaining ground throughout the various metal-mining districts of the United States. The operating companies of the Lake Superior region are for the most part large and strong financially and they can well afford to lead the way in work of this nature. In the Western metal-mining States there is a much larger proportion of small operations and prospects, the owners or lessees of which cannot afford to go to great lengths in the matter of safety.

It is hoped that all mine operators, from the insignificant prospector to the wealthy magnate, will eventually recognize the value of safety work. Sufficient records are now available, both from coal- and metal-mining districts, to prove that safety work pays from every standpoint. In this connection, it is well to be able to say that safety work pays from a financial as well as a humanitarian standpoint, for it is a sad but true commentary that there are still some operators who cannot be appealed to except by a promise of financial gain.

The writer feels that he has not done justice to the subject matter of this paper, especially that part of it dealing with the vast good that has been accomplished through the lessening of death and suffering. Again, some data concerning other phases of the subject have of necessity been omitted owing to the short time that was available in which to prepare this paper.

Gasoline Locomotives in Relation to the Health of Miners

BY O. P. HOOD, PITTSBURGH, PA.

(Pittsburgh Meeting, October, 1914)

NONE of the methods now in use for the transportation of materials underground is entirely free from more or less serious objection. The great flexibility, ease of control and economy of operation of electric tramming are accompanied by the serious menace of a trolley-wire distributing system.

As the gasoline locomotive has even greater flexibility of application and requires no similar dangerous distributing system, it might be considered a safety device which would make possible the elimination of the dangerous trolley system were it not possessed of other objectionable qualities peculiar to itself. It is found that the exhaust gases from the engine may be injurious to the health of those breathing the air in which the locomotive has been operating. While electric shock may kill the individual who makes contact with uninsulated parts, the gasoline engine may be detrimental to the health of all those who have to work within the atmosphere corrupted by exhaust gases. The degree of pollution measures the magnitude of the menace. This may be negligible at times, but with careless operation it becomes serious. The exhaust gases from an engine are composed of nitrogen, a little free oxygen, hydrocarbons, hydrogen, carbon monoxide, and carbon dioxide, the last two being considered dangerous.

The presence of carbon dioxide as a product of combustion of the gasoline was recognized as an objection from the beginning of the use of these machines, but attention was called to the fact that the amount produced was relatively small as compared to that from other sources of this gas, and it was not likely to be made in dangerous quantities. The effect of carbon dioxide, except in relatively large percentages, is confined to the effect it may have in reducing the oxygen content of the air that is breathed.

The presence of carbon monoxide in the exhaust gases in injurious quantities was less apparent, but it appears from what is now known that this is the limiting factor in the use of these locomotives if they are not to be injurious to health. The presence of carbon monoxide in the air in relatively small quantities has been shown to have a marked effect upon the blood, producing sickness, and if inhaled in sufficient quantity it pro-

duces death. A discussion of the effects of these gases is to be found in *Technical Paper No. 62, U. S. Bureau of Mines,* by G. A. Burrell, F. M. Seibert, and I. W. Robertson.

It is difficult to get those most conversant with the effects of carbon monoxide to assign a definite limit to the amount which may be in the air without detriment to health. There does not exist a sharply defined line on one side of which is to be found safety, with danger on the other side. It has been shown that the physiological effects vary with different people, and with the state of health and degree of activity required of the individual, so that a condition which may be considered safe may prove to be unsafe under conditions of harder work and unfavorable physical state. After careful inquiry, the best that can be stated at this time is that, without injury to health, no more than 0.1 per cent. of carbon monoxide can be breathed and that for short and infrequent intervals. It is probable that one-half of this percentage could be allowed for a considerable period of time without noticeable effect.

The percentage of carbon monoxide in the mine air depends upon the amount made by the engine and on the quantity of air with which it is mixed. It will be necessary to provide ventilation for the worst possible combination of gases which such engines can make under unskillful handling, or else to become informed as to the actual amount of carbon monoxide produced and provide air accordingly. To provide for the worst conditions it becomes necessary, then, to consider the gasoline locomotive as a carbon monoxide producer, and to discover the maximum quantity of this gas which any engine is capable of making. It is not sufficient to consider the average amount produced as distributed over the whole time of running such a machine. The total quantity of gasoline burned in any one day may have produced but a small quantity of carbon monoxide, but if this has been confined to a relatively short period during bad carburetor adjustment, and in some poorly ventilated space, the momentary percentage may be very high and the consequences may be fatal. It is evident that to be entirely safe the ventilation must be sufficient to keep the percentage of carbon monoxide below the assigned limit when the engine is producing the maximum quantity possible. If this maximum quantity is provided for by proper ventilation, the chance of injury to health can be considered to be quite remote. Certain peculiarities of gasoline engines make the percentage of carbon monoxide generated vary between rather wide limits, but the maximum is fairly constant. No other constituent of the exhaust gases varies so much or so rapidly with slight changes of adjustment as does the carbon monoxide.

Extended experiments have been made by the Bureau of Mines to determine under what conditions of running such locomotives give a maximum quantity of carbon monoxide. This maximum quantity is found when the engine is running at full load and full speed, and when

burning the greatest quantity of gasoline which will maintain these conditions. Any further enriching of the mixture results in weaker explosions, misfires, a reduction in speed of the locomotive, and thus a reduced total quantity of gases. The weight of gasoline used per stroke may vary nearly 100 per cent. without varying the speed or power. As the mixture is enriched the combustion is less complete, reducing the efficiency but affecting the power but little until the upper limit of explosibility is approached, when the power falls rapidly. Under these conditions a maximum quantity of carbon monoxide equivalent to $5\frac{3}{4}$ per cent. of the piston displacement may be generated and delivered into the surrounding air. When the engine is throttled down to a slow speed and with a light load it is possible to produce exhaust gases containing a greater percentage of carbon monoxide but the total quantity is less because of the slow speed of the engine. It can therefore be stated that the amount of carbon monoxide possible to generate with a gasoline locomotive is a function of its piston displacement.

TABLE I.

Engine Cylinder Size, In.	No. Cylinders	Speed Rev. per Min.	Piston Displacement *a* (Cu. Ft. per Min.)	Maximum Probable Amount of Noxious Gases (Cu. Ft. per Min. at 60° F. and 30 in. Barometer) Produced with				Amount of Air (Cu. Ft. per Min.) Required to Dilute Exhaust Gases to 1 Part CO per 1,000 Parts of Air *b*	
				Good Carburation		Bad Carburation		Good Carburation	Bad Carburation
				CO	CO_2	CO	CO_2		
4.75 by 5.25	4	800	172	2.61	6.80	9.91	3.65	2,610	9,910
5 by 5	4	600	136	2.06	5.37	7.84	2.88	2,060	7,840
5 by 5	4	800	182	2.76	7.18	10.48	3.86	2,760	10,480
5 by 6	4	800	218	3.30	8.60	12.56	4.62	3,300	12,560
5.5 by 5	4	600	165	2.50	6.51	9.50	3.50	2,500	9,500
6 by 6	4	700	275	4.17	10.86	15.85	5.82	4,170	15,850
6 by 7	4	500	229	3.47	9.04	13.19	4.85	3,470	13,190
6.5 by 7	4	500	269	4.07	10.63	15.50	5.70	4,070	15,500
6.5 by 8	4	650	399	6.04	15.76	23.00	8.46	6,040	23,000
7 by 7	4	500	312	4.73	12.33	17.97	6.62	4,730	17,970
7 by 7	6	500	468	7.08	18.49	26.97	9.92	7,080	26,970
8 by 7	4	500	407	6.16	16.08	23.45	8.62	6,160	23,450
8 by 7	6	500	610	9.24	24.10	35.14	12.93	9,240	35,140

a Area piston in square feet × stroke in feet × number of cylinders × revolutions per minute.

b Maximum amount of carbon monoxide which can be breathed for short and infrequent intervals without injurious effects.

The sizes of locomotives in common use are given in the accompanying table, together with the maximum quantity of injurious gases which they may generate.

The size of a locomotive that may with safety be introduced into a mine depends upon the amount of air that can be mixed with the exhaust gases in the most unfavorable portion of the run of the locomotive. For each cubic foot of carbon monoxide possible to generate in the engine there should be available 2,000 cu. ft. of air to mix with the exhaust gases if this air is for continued breathing, while for short and infrequent intervals the proportion may rise to one part in one thousand.

The air with which the exhaust gases are mixed is the air passed through by the locomotive. If the speed of the air current and that of the locomotive are the same there would be a concentration of exhaust gases immediately surrounding the engine, which would very soon reach the danger limit. The engine runner would be exposed to this excessive amount. Or, the air may be stationary, and if the locomotive passes though this air at such a rate as to distribute the exhaust gases into the required quantity of air the dilution may be below the danger limit. It becomes necessary, therefore, to investigate the relative velocity of the locomotive and the air current, together with the cross section of this air current, in order to determine whether a sufficient quantity of air is passed to insure a proper dilution. The mixing due to the movement of the cars and the diffusion of the gases is expected to distribute the gases through the air. An investigation of this kind will frequently disclose some portion of the run of the locomotive where the air and the locomotive are traveling together and the dilution is insufficient, forming the basis for just complaint. By a change in the speed of the locomotive, or more careful running, perhaps with lighter loads, these difficulties can frequently be overcome. It is evident from the foregoing that it is a dangerous practice to allow locomotives to stand idle for any length of time in locations where there is not ample movement of air. The introduction of a self-starter on locomotives which are compelled frequently to wait for cars would be a great aid. The engines on these locomotives are usually difficult to start, and to stop the engine whenever the locomotive is compelled to wait for a few minutes would not be practicable unless some easy means of re-starting was provided, but this practice would greatly reduce the chance of air pollution.

The requirement that not less than 1,000 cu. ft. of air shall pass the locomotive for each cubic foot of carbon monoxide possible for the engine to generate will limit the size of the locomotive in many cases to sizes smaller than are now in use. It may also confine the use of these engines to main entries where there is ample ventilating current.

This maximum quantity of noxious gases which is here provided for should not actually be realized with improved carburetor construction

and skillful manipulation. The percentage of carbon monoxide in the exhaust gases under the worst conditions is about 13½ per cent., while under normal running conditions the amount seldom exceeds 6 per cent. It is this great variation in the possible amount of carbon monoxide due to unskillful operation that makes the experience of different users of these machines vary through the wide limits noted. If a locomotive is installed in a place where the quantity of air is not sufficient properly to dilute the maximum quantity of carbon monoxide, analyses of the exhaust gases should be made to determine just what percentage is actually produced with the carburetor in use and the methods employed in running these machines. The ventilating current may then be regulated to conform to the actual amount of carbon monoxide as found. It is proposed that this be done by a method somewhat as follows:

Suppose, upon analysis, that the actual amount of carbon monoxide is found to be not over 4 per cent.; allowing a factor of safety of 2, and assuming that this percentage might on occasion mount to 8 per cent., and, since the maximum percentage possible is about 13½ per cent., the ventilating current required can be regulated in the proportion of 8 to 13½.

If a careful study is made of the amount of air available for dilution throughout the run of a locomotive, and the quantities are kept within the limits that are here proposed, a wide field of usefulness for these locomotives remains, where it is believed there would be no injurious effects upon health.

Discussion

R. V. Norris, Wilkes-Barre, Pa.—I have had a little experience with gasoline locomotives and I want to indorse very strongly Mr. Hood's point with regard to the danger of carbon monoxide poisoning. I think the terrible danger is not fully realized.

I was foolish enough some time ago, in investigating a gasoline locomotive, to attempt to walk out of the mine entry behind it, and for several days I felt very seriously the carbon monoxide effects.

The use of gasoline locomotives ought to be confined to mine entries where there is a very large amount of air. One of the grave dangers is their great convenience and the temptation to run them into chambers for just a few moments to get a car or two.

George S. Rice, Pittsburgh, Pa.—Apparently one of the great dangers is in the style of the gasoline engine. When my attention was first directed to the type of gasoline locomotives which I saw in the foreign mines—France, Belgium, and Germany, I was most favorably impressed with them, and they have given no trouble on account of fumes, so far as I could learn. Large numbers of them were being used, but all were of small capacity, and of a kind in which the speed control is by a governor of the

hit-or-miss type, the engine running continuously. Their power capacity was very low, if I remember correctly it was not over 4 or 5 h.p., but they were used under favorable conditions where the grades in the mines were slight, since the entries or tunnels had a slight grade in favor of the load, making about the same pull for the empties against the grade as for the load with the grade. I understand the hit-or-miss valve arrangement enables a setting that will give more perfect combustion than can be insured with the valve controlled by the operating man, as in the automobile type of valve and engine used in this country. Apparently the European type of locomotive does not have the capacity to permit use under the conditions existing in this country, where most of our entries or tunnels have variable grades and long heavy trips of mine cars are used.

S. A. Taylor, Pittsburgh, Pa.—I have had a little experience along this line in a mine of low vein coal where just such conditions as Mr. Rice speaks of were prevalent. We had great difficulty with the motor and did not use it except on return-air courses, using our main heading as the return-air way. We had no good method of regulating the amount of gasoline admitted into the engine. This seems to be one of the weakest points in all of the gasoline locomotives I have examined. When the motor stalls or checks up a little the man running the motor will throw in excessive gasoline, and the fumes are extremely hard to handle even with a very good current of air. In order to overcome the stalling we took up the track and relaid it to a uniform grade, but even then, when there was an extra wagon—a ton or two on a trip—we would find the same difficulty arising. We operated our fans and pumps with electricity. On one occasion the generator broke down, and while we were waiting for a piece to replace the broken part the sump began to fill up; as the pump was in there the motor was run in quickly to pull it out, with disastrous results. Two men were working at the pump. No air current was passing, and when the motorman started the motor a lot of gasoline was turned on, with the result that one man was overcome almost instantly, requiring several minutes' work to relieve him, and our mine foreman, who was not nearly so susceptible to the fumes, felt the effects very perceptibly. One of the main things, I believe, is to get a device on the motor that will control in a very much better way the amount of gasoline admitted into the engine.

Another thing of importance is to employ capable men to run it. In these days of automobiles, you would think that would not be a very serious question, but it is around a mine. You can put on one of your best machinists to run the motor; if it stalls, he gets rattled about his machine not working, and if he has no automatic control he will throw on too much gasoline, and trouble results.

W. G. Whildin, Lansford, Pa.—We had an experience of more than a year with five gasoline locomotives, 10-ton type, of three different

makes. We found that it was not so much a question of the health of the miners, because the men were not affected at all, but the health of the company's finances, which at that particular plant suffered very much. We finally put in an electric haulage system which we had been religiously keeping away from on account of its greater cost.

After the installation of the electric haulage system, about four months ago, we found our output increased 30 per cent. from exactly the same openings.

The main trouble with the gasoline locomotives was that they were not dependable. In the morning all five of them would start out on their runs in fine condition, and sometimes—in fact, very often—within a few hours, three and sometimes four of them would be out of service, and trips stalled in all parts of the mine. As a result, the locomotives which were still working were overloaded by having to move these cars and their regular trip in order to clear the road.

As I stated, our men were not bothered at all by the fumes from the locomotives. Our workings are all on heavy pitches, and by actual tests men working in blind ends were not affected even when the motors were located directly below them. We had one case, however, where four motormen were so badly overcome that the use of the pulmotor had to be resorted to. This occurred at a point near the foot of the shaft where we had the locomotives stored over night, and when the engines were started in the morning the fumes were very strong.

So far as care of the locomotives was concerned; I want to say that at all times they received the best care possible. Every night two high-class repairmen, assisted by some or all of the motormen as was found necessary, overhauled the locomotives.

O. P. Hood, Pittsburgh, Pa.—Since the discussion has been confined to the difficulties that have been met with, I feel like defending the gasoline locomotive. The great difficulty has been that people do not recognize the presence, the source, or the magnitude of the danger in using these machines.

It would be a bad thing to place powder in the hands of those who do not know its danger, but simple rules and regulations allow its extensive use in proper hands. The same can be said of gasoline locomotives. That there is danger from the exhaust gases is not recognized until after some unfortunate experience such as has been related in the discussion. The limits are not known—just how far one can go and still be safe—and for this reason investigations were made by the Bureau of Mines. It is believed that under proper regulations and with intelligence the machines can be safely used. The gasoline locomotive can be easily overloaded and evidently with disastrous results, as related, but some regulation resisting this tendency should not be hard to enforce when it is made evident that it is necessary for safety.

The makers of the gasoline motor are of the opinion that a really good carburetor such as you would expect to have on your automobile is not suited for mining machinery. Gasoline locomotives use but one make of carburetor, its great advantage being ruggedness and simplicity. Five or six per cent. of carbon monoxide may be made with these carburetors, while in certain long-distance economy runs of automobiles with better carburetors the exhaust gases contained only a fraction of 1 per cent. This shows what it is possible to do with gasoline locomotives.

The gasoline engine in a mine should be frankly recognized as a possible source of danger and the necessary conditions for safety provided. When the danger is thus recognized the conditions for safety are not difficult of attainment in many places.

A Test of Centrifugal Motor-Driven Pumps

BY S. S. RUMSEY* AND W. F. SCHWEDES,† DULUTH, MINN.

(Pittsburgh Meeting, October, 1914)

Introduction

In order to realize the economies which would result in operating the mine pumps at the Chapin mine with electricity generated by water power, the Oliver Iron Mining Co. recently installed a hydro-electric plant at the upper Quinnesec falls on the Menominee river, and constructed a transmission line to the mine.

On account of the large flow of water in the Chapin mine, and the great depth from which it has to be handled, it was necessary to make a thorough investigation of the merits of both the motor-driven centrifugal and motor-driven plunger pumps, in order that the most economical results as regards power, cost of installation, maintenance, and operation, be obtained. This investigation resulted in the selection of motor-driven centrifugal pumps, as the cost of the pumps and motors, installation, and maintenance was considerably lower, and the space required in the underground workings very much smaller, than for the plunger pumps. The overall efficiencies guaranteed by several of the centrifugal-pump manufacturers were so nearly those guaranteed by plunger-pump manufacturers that the saving in power which would result from the installation of plunger pumps did not justify their increased first cost and the increased cost of installation under the conditions prevailing at the Chapin mine.

In order to substantiate the representations of the centrifugal-pump builders it was deemed necessary to insist upon a factory test of the completed units before shipment, the test to be conducted under conditions of head and capacity approximating as nearly as possible those under which the pumps would operate in service. Furthermore, it was considered necessary to demonstrate, by test, that there were no deficiencies in design or capacity of the pumps or motors before their final installation in the mine, as it would be extremely difficult to make alterations in the pumps or motors after they were installed.

* Chief Engineer, Oliver Iron Mining Co.
† Electrical Engineer, Oliver Iron Mining Co.

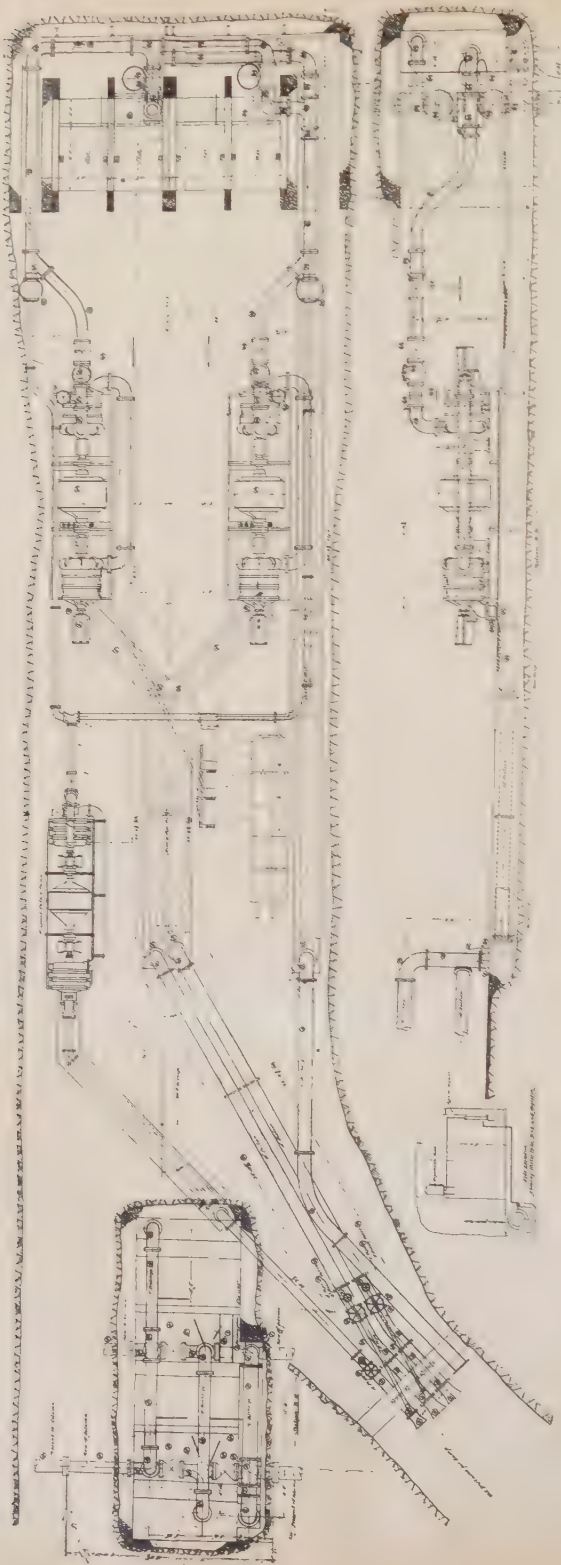


FIG. 1.—PLAN AND SECTIONS OF THE 12TH LEVEL PUMP STATION, NO. 2 HAMILTON SHAFT, CHAPIN MINE.

The general arrangement of the pumps, piping, and electrical control for the 12th level is shown in Fig. 1, and for the 16th level in Fig. 2.

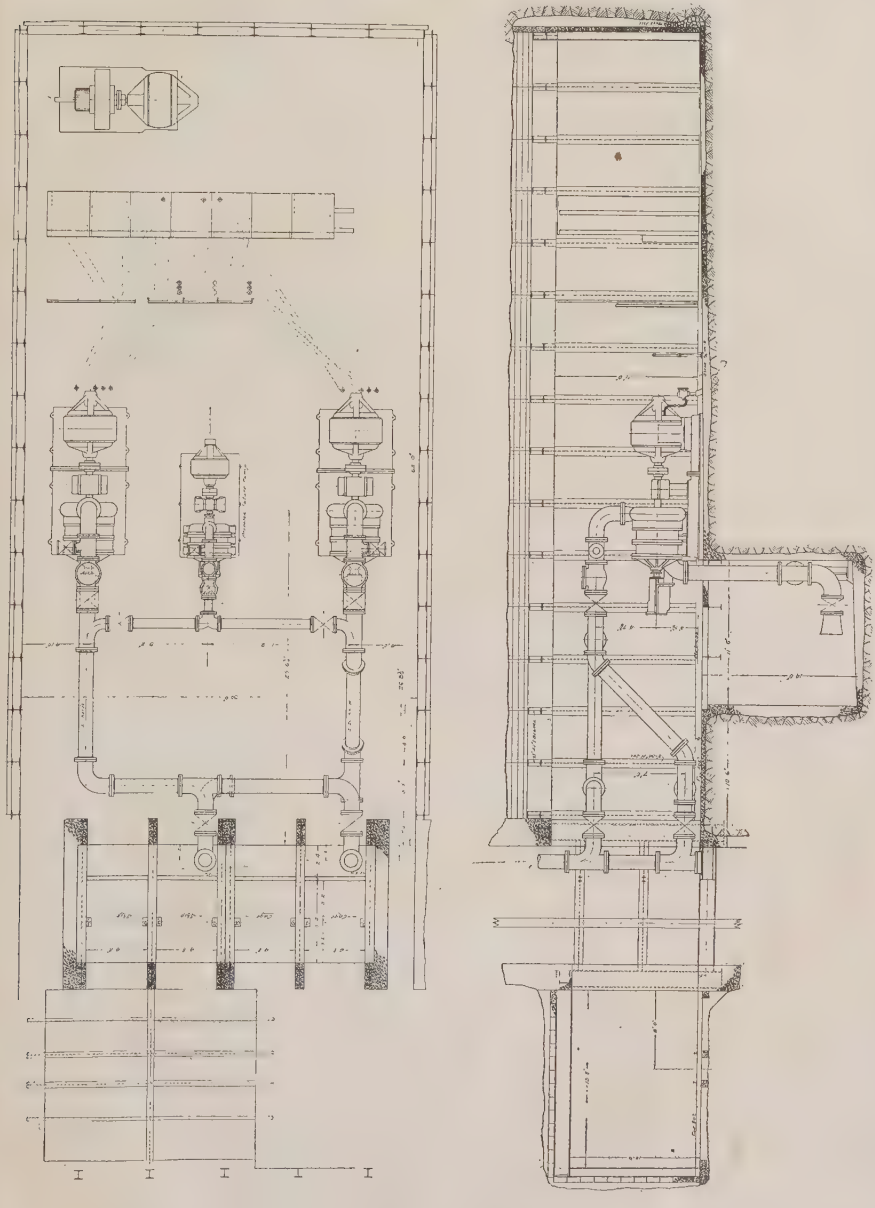


FIG. 2.—PLAN AND SECTION OF THE 16TH LEVEL PUMP STATION, NO. 2 HAMILTON SHAFT, CHAPIN MINE.

Power is transmitted from the surface to 12th-level pumps through two 250,000-circular mil, three-conductor, 30 per cent. Para rubber-insulated, lead-covered, steel-armored cables; and from the surface to 16th

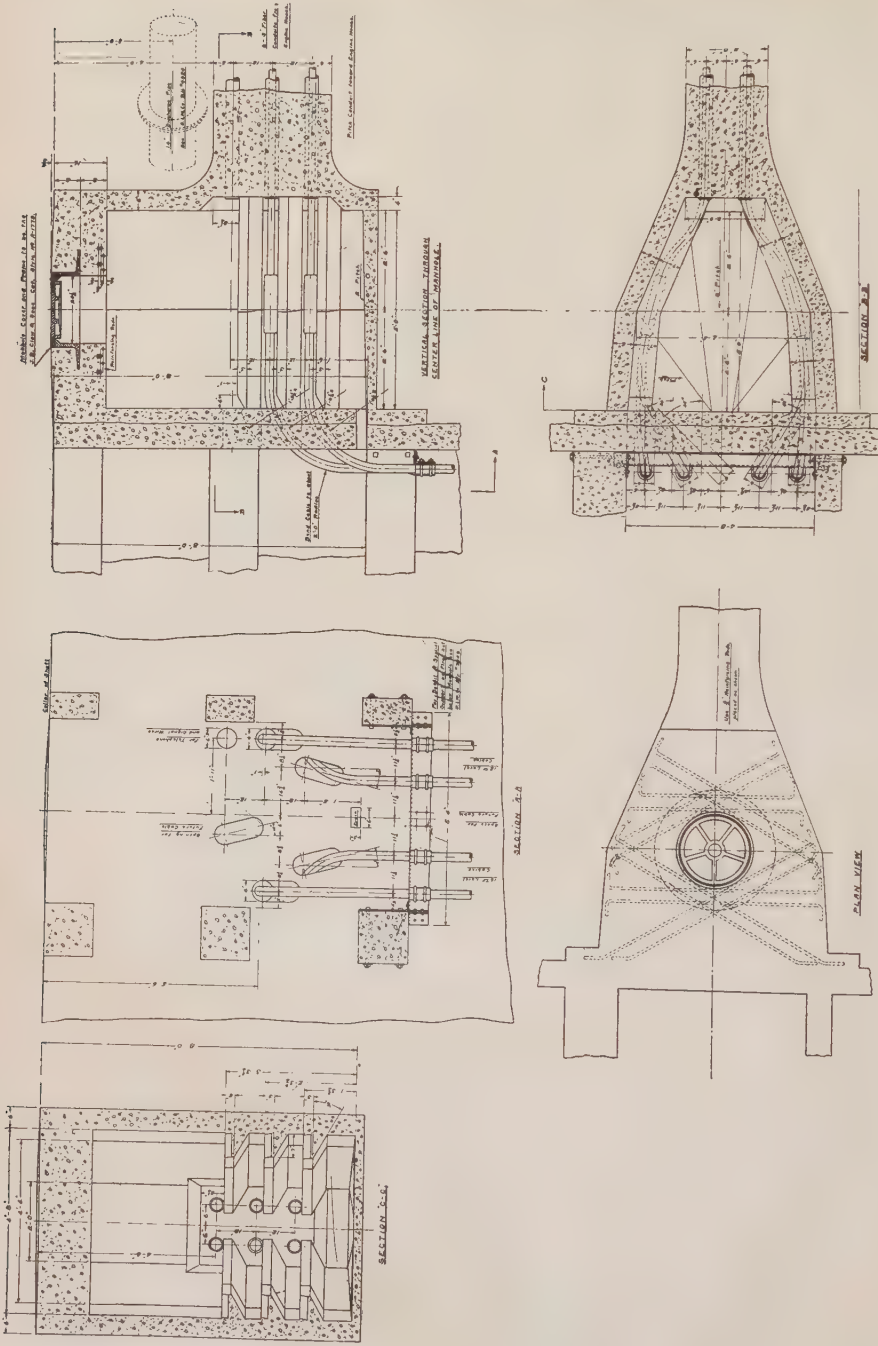


FIG. 3.—DETAILS OF CABLE MANHOLE FOR 2,300-VOLT CABLES.

level through two 3-0 three-conductor, 30 per cent. Para rubber-insulated, lead-covered, steel-armored cables. These cables were manufactured by the American Steel & Wire Co., in conformance with specifications which were especially suited to the conditions.

The method of carrying these cables down the vertical mine shaft, and supporting them by the concrete shaft timbers, is shown in Fig. 3.

Specifications for all equipment required for the power and pumping plants, and all plans for the installation of same, were prepared in the office of the Chief Engineer of the Oliver Iron Mining Co., at Duluth, Minn.

Report of Test

In accordance with the contract with Henry R. Worthington Co., tests were run on the four 3,000-gal., motor-driven centrifugal pumps built by them for service on the 16th and 12th levels, No. 2 Hamilton shaft, Chapin Mine, Iron Mountain, Mich. The complete erection and testing of these four motor-pump units was witnessed by the Electrical Engineer of the Oliver Iron Mining Co. The first pump was turned over Dec. 13, 1913, and the test completed on the fourth pump Jan. 13, 1914.

Description of Motor-Driven Pumps

The water to be pumped from the Chapin mine is encountered on the 16th level, and is approximately 3,000 gal. per minute. The vertical lift from the sump on the 16th level to surface will be 1,358 ft., and the water will be pumped to the surface in two lifts, the pumps being located as described below:

Two 3-stage pumps will be installed on the 16th level, each having a capacity sufficient to handle the entire mine flow to a sump on the 12th level, a vertical lift of 392 ft. The estimated dynamic head against which each 16th-level pump will operate when throwing 3,000 gal. of water per minute is 412 ft.

Two 6-stage pumps will be installed on the 12th level, each having a capacity sufficient to handle the entire mine flow to surface, a vertical lift of 966 ft. The estimated dynamic head against which each 12th-level pump will operate when throwing 3,000 gal. of water per minute is 1,000 ft.

Under normal conditions only one pump on the 16th level and one pump on the 12th level will operate simultaneously. There are, however, two independent discharge column pipes, making it possible to operate both pumps on both levels simultaneously, thereby providing a maximum capacity of 6,000 gal. per minute from the 16th level to the surface.

Fig. 4 shows the general arrangement of present steam and air pumps and new electrical pumps.

The pumps are of the multistage turbine type, built in units of three

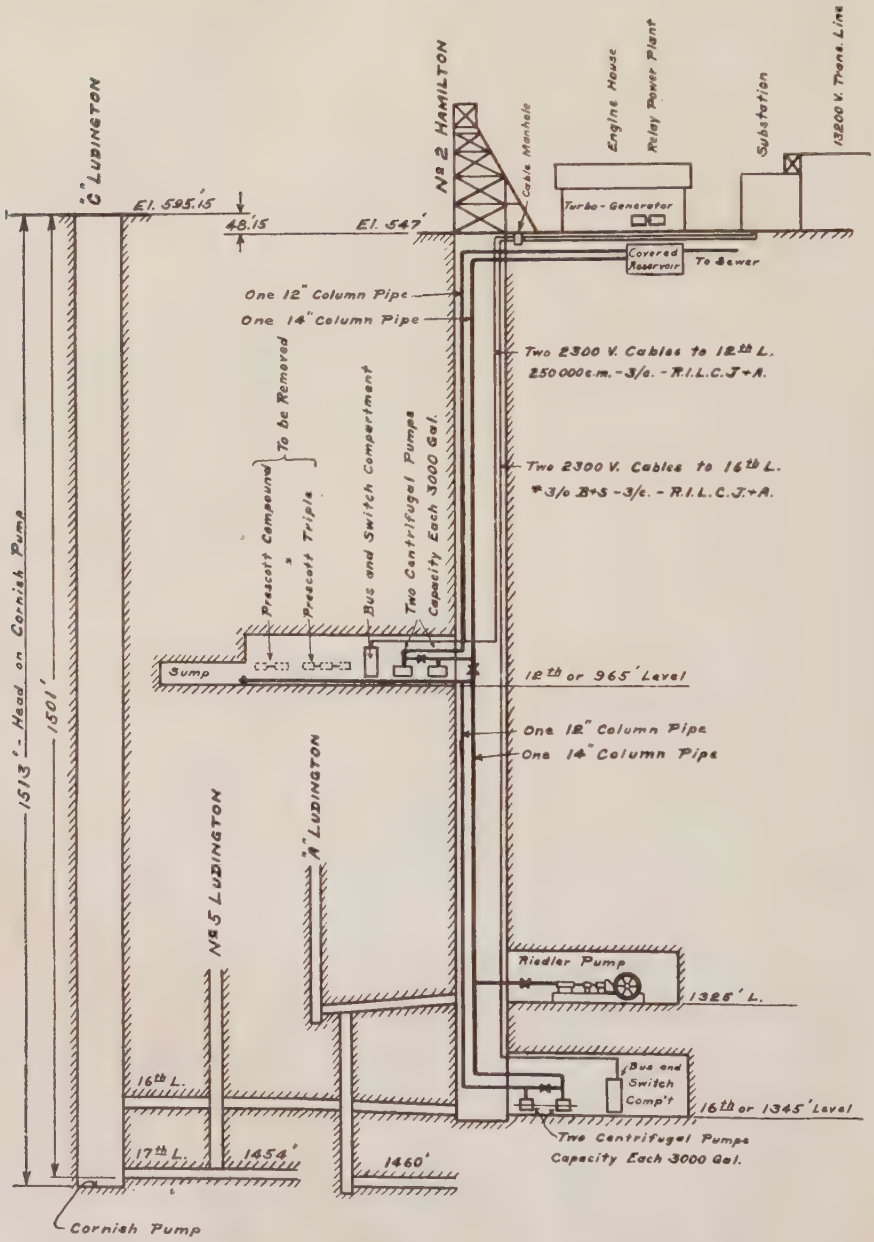


FIG. 4.—PUMPING DIAGRAM, SHOWING STEAM AND ELECTRIC PUMPS.

stages each. The two 16th-level pumps each consist of one unit of three stages, mounted on a common cast-iron bedplate, with a 450-h.p. 1,200 rev. per minute induction motor direct connected through a flexible coupling. The 12th-level pumps each consist of two three-stage units, mounted on a common cast-iron bedplate, and direct connected through a flexible coupling to each end of a 1,050-h.p. 1,200 rev. per minute induction motor, the discharge of the low-pressure unit being connected through a 12-in. pipe to the suction end of the high-pressure unit.

The three-stage units that make up the four pumps are identical in design and dimensions so that all parts are interchangeable, excepting the impellers in the six-stage pumps, which have a trifle larger diameter as their operating head is more than twice that of the three-stage pumps.

The pumps are designed to allow lowering through a vertical mine shaft 3 ft. 10 in. wide by 6 ft. 4 in. long. All glands are water sealed and all bearings are water cooled; casings are cast steel; impellers and diffusion vanes are bronze.

Each pump is driven by a General Electric 2,200-volt, three-phase, 60-cycle, 1,200 rev. per minute, wound secondary induction motor, with provision for short-circuiting secondary windings at the control board by means of an oil switch which is so interlocked with the drum controller and primary circuit breaker that the entire control equipment is as "fool-proof" as is possible to make it.

All the 2,200-volt control equipment will be mounted in concrete compartments and all wiring around motors will be so protected that all possible danger to operator is eliminated.

These centrifugal pumps will replace the present steam pumping equipment. The rating of the centrifugal pumps is shown in Table I.

TABLE I

	12 in. 3-Stage 16th Level	12 in. 6-Stage 12th Level
Capacity in U. S. gallons per minute	3,000	3,000
Total dynamic head in feet	412	1,000
Rev. per minute at full load	1,178	1,170
Diameter of suction in inches	12	12
Diameter of discharge in inches	12	12
Horsepower of motors	450	1,050
Electrical characteristics of motors: 2,200 volts, three-phase, 60 cycles.		

Purpose of Test

Each pump with its motor was tested to determine: (1) Head-capacity curves of pumps; (2) overall efficiency, wire to water; (3) degree

with which builder's guarantees were met; (4) mechanical operation of pumps and motors for vibration, thrust, heating, and noise.

Method of Testing

The four pumps were shipped from the Worthington factory at Harrison, N. J., to the factory of the General Electric Co. at Schenectady, N. Y., and each complete unit assembled and aligned on a steel floor plate.

Power supply for pumps Nos. 1, 3, and 4 was taken from a 12,500-kva., 2,200-volt turbo-generator which was run especially for this test.

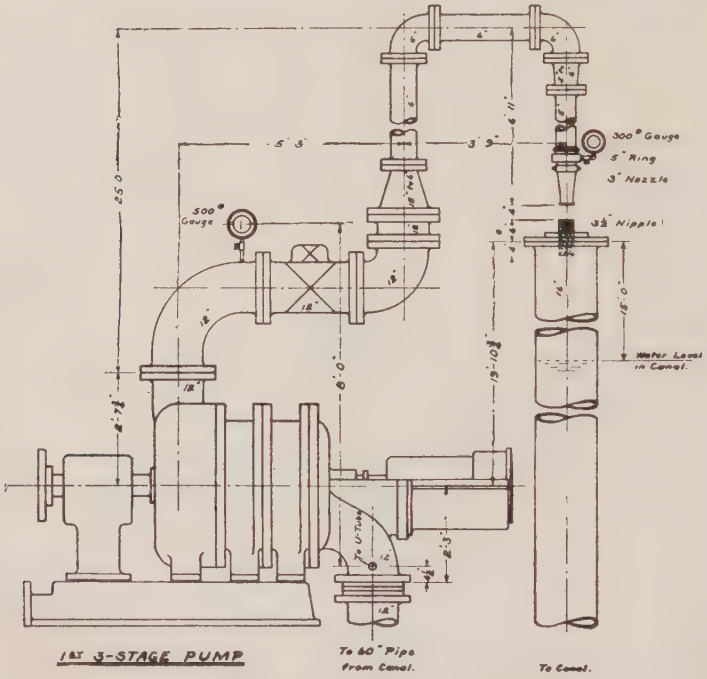


FIG. 5A.—ARRANGEMENT OF DISCHARGE WATER PIPING OF FIRST THREE-STAGE PUMP.

Pump No. 2 was supplied from a 750-kw. direct-current motor-driven alternating-current generator. Constant frequency was available from the turbo-generator, but the frequency of the motor-generator set fluctuated and test readings were taken as frequency became reasonably steady at 60 cycles. Power input to motors was measured with previously calibrated electric instruments. The head on pump was obtained by means of a pressure gauge between pump and discharge valve and a U-tube in pump suction. The pump discharge in gallons per minute was measured with a calibrated 3-in. nozzle. Figs. 5A, 5B, and 5C shows the

piping around pumps, as set up for each test, and the location of pressure gauges and U-tube. All pressure gauges were carefully calibrated and all pressure readings in the following data are corrected.

The 3-in. nozzle was calibrated over a rectangular weir up to 46½ lb. nozzle pressure, and Table II gives readings taken on this calibration.

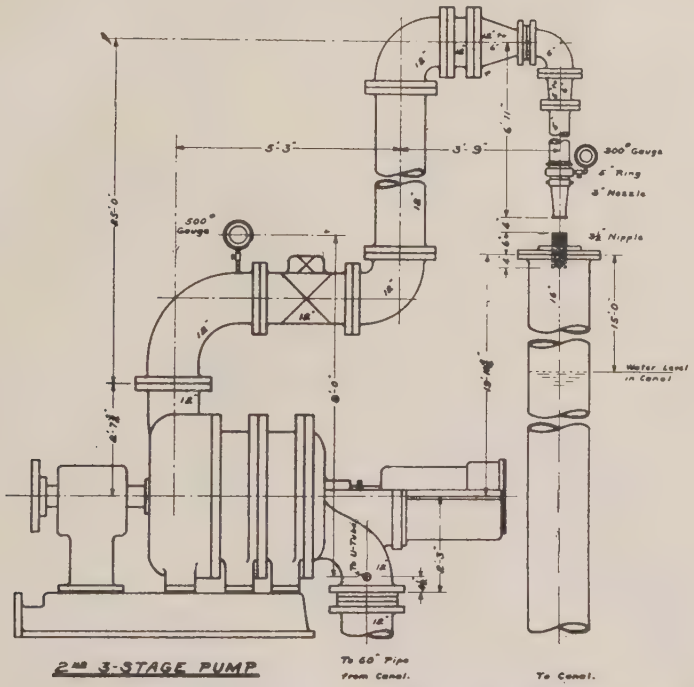


FIG. 5B.—ARRANGEMENT OF DISCHARGE WATER PIPING OF SECOND THREE-STAGE PUMP.

TABLE II

Pressure on Nozzle *P*	Gallons per Minute over Weir *C*	Constant *K*
7⅜	820	91,200
13⅜	1,105	91,500
19½	1,330	90,750
26	1,520	89,000
29¼	1,610	89,000
41½	1,900	87,100
46½	2,040	89,400

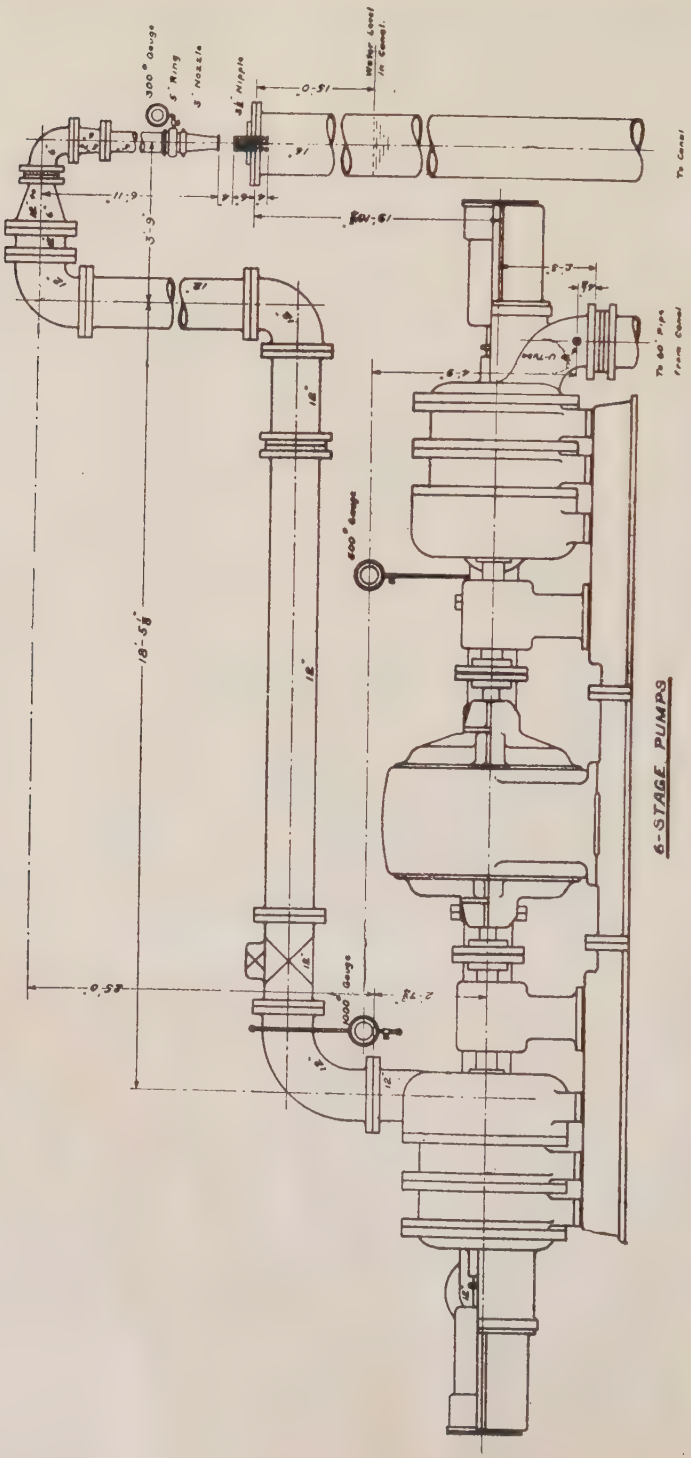


FIG. 5C.—ARRANGEMENT OF DISCHARGE WATER PIPING OF SIX-STAGE PUMPS.

From the readings in Table II, the average value of constant K = 90,000. Substituting in equation $C^2 = KP$, the pressure-capacity curve shown in Fig. 6 was prepared. Although there is a slight variation in the value of constant K for different nozzle pressures, as shown in Table II, this does not appreciably affect values for gallons per minute discharge, as C varies as the square root of K.

Water was pumped from a 60-in. pipe-line connection from the Erie Canal and the 3-in. nozzle discharged into a 16-in. stand pipe back to canal. As the level of water in the canal was higher than the suction

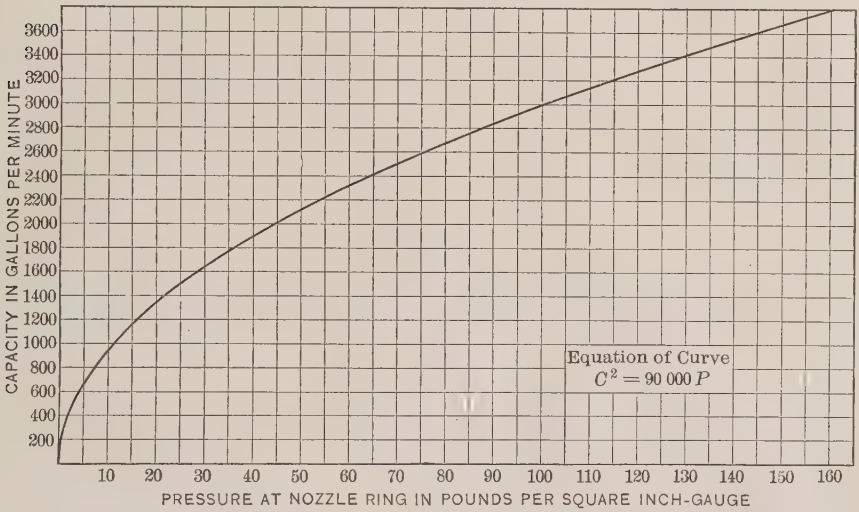


Fig. 6.—Pressure-Discharge Curve for 3-in. Nozzle and 5-in. Ring.

connection on pump, the pressure on suction varied from 2 to 6 in. of mercury and was measured with the mercury U-tube. (See Figs. 5A, 5B, and 5C for location of tube connection in pump suction.)

In taking the test readings on each pump, no preliminary test runs were made or readings taken, and each pump was operated before the test for only a sufficient time to inspect piping and assure proper lubrication, tight glands, and true alignment.

Test Data

In the following tables, pumps are numbered as follows: No. 1, 16th level, 12 in., 3-stage, 450 h.p.; No. 2, 16th level, 12 in., 3-stage, 450 h.p.; No. 3, 12th level, 12 in., 6-stage, 1,050 h.p.; No. 4, 12th level, 12 in., 6-stage, 1,050 h.p.

Table III gives the starting current for each pump. These were taken with the 12-in. discharge valve closed and the 2-in. bypass valve open, which will approximate conditions of starting when installed.

The motors were started with their respective resistance grids and controllers. The starting duty is so light that no trouble should be expected with the drum controllers.

It will be possible to start these pumps within 25 sec. without exceeding full-load current, by using a greater rate of acceleration on the first number of notches on the controller and a decreasing rate of acceleration on the last few notches, as the pump takes its load.

Motor efficiencies and power factors for one 1,050-h.p. and one 450-h.p. motor were obtained by separation of losses.

In Table IV are shown the guaranteed and obtained efficiencies and power factors for the two motors.

Tables V, VI, VII, and VIII for pumps 1, 2, 3, and 4, respectively, give obtained data from which curve sheets (Figs. 7, 8, 9, and 10) were prepared, giving head-capacity curves, horse-power input curves, and overall efficiency curves. The pump-efficiency curves were obtained by dividing overall efficiency by motor efficiency.

Test Results

Table IX was prepared from the curve sheets to show the degree with which the builder's guarantees were met.

Motor Slip

On pumps Nos. 3 and 4, the motor slip was measured directly by means of a 5-volt voltmeter connected across the slip rings, which gave actual values. The percentages of slip given in Tables VII and VIII are calculated, assuming 60-cycle power supply. Inconsistency in slip values is due to slight variations in speed of the prime mover, which it was impossible to check within the limits of error, in the values of slip.

Slip for pumps Nos. 1 and 2 was obtained by means of speed counter and tachometer and the figures are not accurate, but indicate maximum values that can be expected.

When in actual service, the slip for all motors will be less than any values given in data sheets, because of the long secondary leads used on test.

Temperatures, Motors, and Bearings

Because of the large amount of power and time required to obtain final full-load temperatures on motors and bearings, no attempt was made to obtain these readings further than those taken during test, which in each case covered a period of about 2 hr.

The highest temperatures recorded at the end of test runs were in the end turns of stator windings given by thermometers Nos. 3 and 4.

Table X shows maximum temperature rises above 25° C. at end

TABLE III.—*Starting Currents*

Controller Notches	Pump No. 1 Amperes	Pump No. 2 Amperes	Pump No. 3 Amperes	Pump No. 4 Amperes
S	43	79	150	157
1	43	49	160	143
2	47	49	150	126
3	51	51	110	110
4	61	57	110	118
5	65	53	110	118
6	67	61	140	118
7	79	69	140	118
8	69	79	140	118
9	77	79	140	118
10	79	83	140	126
11	75	84	120	126
12	89	96	160	157
13	98	118	190	197
14	89	79	150	118
Total time, seconds	30	25	25	30
Full-load motor amperes.	101	101	229	229

TABLE IV

	Loads			
	½	¾	Full	1¼
Guaranteed efficiency, 450-h.p. motor.....	86.5	91.	92.	92.5
Obtained efficiency, 450-h.p. motor.....	92.4	93.9	94.65	94.9
Guaranteed efficiency, 1,050-h.p. motor.....	90.5	92.5	94.	94.
Obtained efficiency, 1,050-h.p. motor.....	93.	94.4	95.	95.2
Guaranteed power factor, 450-h.p. motor.....	82.	88.5	92.	93.
Obtained power factor, 450-h.p. motor.....	86.1	90.3	91.3	91.5
Guaranteed power factor, 1,050-h.p. motor...	90.	92.	93.5	93.5
Obtained power factor, 1,050-h.p. motor.....	90.8	93.2	93.3	93.3

of test runs and gives an idea of what can be expected of these motors under continuous operation.

Being high-speed motors, a relatively small amount of material entered into their construction, giving them a small thermal capacity, so that the above temperature rises are nearly final. The guaranteed full-load temperature rises for 1,050-h.p. and 450-h.p. motors is 35° C.

TABLE V.—*No. 1 Pump Unit for 16th Level Station*

Time	Motor							Pump							Efficiencies					*Temperatures				
								Head-feet					Capacity											
	Volts	Amperes	Kv.A.	Kw.	Power Factor	R.P.M.	Per Cent. Slip	Mercury Column Suction.	Ft. S	Gauge Pressure Discharge	Ft. D	D−S+8 ft. Total	Gauge Pressure Nozzle	Gal. per Min.	Water Hp.	Motor Hp. Input	Wire to Water Efficiency.	Motor Efficiency	Pump Efficiency	1	2	3	4	5
A.M.																								
10.55	2,180	64.8	244	224	91.8	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	24.7	24.0	26.0	25.0	28.0
11.10	2,190	102.0	387	356	92.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	25.0	24.0	34.0	33.0	44.0
11.25	2,190	101.0	384	351	91.5	1,190	0.83	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	24.5	23.5	41.0	39.4	48.5
11.40	2,190	101.0	384	351	91.5	1,190	0.83	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	24.5	24.5	44.0	42.0	51.2
11.55	2,190	101.0	384	351	91.5	1,190	0.83	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	25.0	22.0	45.5	43.0	53.0
P.M.																								
12.10	2,190	101.0	384	351	91.5	1,190	0.83	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	24.0	22.5	46.0	43.0	54.0
12.25	2,185	108.0	409	379	92.7	1,185	1.25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	24.0	22.0	47.0	44.5	54.0
12.40	2,185	107.0	406	378	93.0	1,185	1.25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	23.8	22.0	47.0	45.0	54.0
12.45	2,178	108.7	410	377	91.8	1,181	1.58	2.0	2.3	183.5	423	429	128.4	3,400	369	505	73.0	94.7	77.1	24.0	22.0	48.0	45.0	54.0
1.00	2,185	107.9	409	377	92.2	1,186	1.17	2.0	2.3	183.5	423	429	128.4	3,400	369	505	73.0	94.7	77.1	23.5	24.0	49.0	45.5	55.0
1.10	2,178	104.0	392	363	92.6	1,182	1.50	2.0	2.3	191.5	441	447	114.0	3,205	362	485	74.6	94.7	78.8	25.0	25.0	49.5	47.0	55.5
1.25	2,184	103.6	392	363	92.6	.....	.....	2.0	2.3	190.5	439	445	114.0	3,205	361	485	74.4	94.7	78.5	25.5	24.5	50.0	47.0	56.0
1.40	2,198	102.8	391	354	90.6	1,182	1.50	2.0	2.3	195.5	450	456	101.8	3,025	349	474	73.7	94.65	77.9	25.5	24.5	49.5	46.5	56.5
1.50	2,188	102.4	388	353	91.0	.....	.....	2.1	2.4	196.0	452	458	100.8	3,015	350	473	74.1	94.65	78.3	.....	.....	.....	.....	.....
2.00	2,198	95.0	361	331	91.6	1,189	0.92	2.5	2.8	199.0	459	464	82.5	2,725	321	443	72.5	94.5	76.7	25.5	23.5	48.5	45.5	57.0
2.20	2,188	95.4	362	331	91.4	1,192	0.67	2.5	2.8	199.0	459	464	82.5	2,725	321	443	72.5	94.5	76.7	25.5	25.0	47.5	44.5	56.5
2.25	2,198	88.3	336	309	92.0	.....	.....	3.0	3.4	203.5	469	474	57.5	2,275	273	414	66.0	94.2	70.1	.....	.....	.....	.....	.....
2.35	2,198	81.2	309	280	90.7	1,195	0.42	3.5	4.0	212.0	488	492	36.6	1,815	226	375	60.2	94.0	64.0	25.5	25.0	46.0	43.5	57.0
2.40	2,198	72.6	276	250	90.6	.....	.....	3.5	4.0	218.0	502	506	17.5	1,255	161	336	47.9	93.8	51.0	.....	.....	.....	.....	.....
2.45	2,201	65.6	250	223	89.2	.....	.....	4.0	4.5	225.0	518	521	0.0	0	0	299	0.0	93.5	0.0	.....	.....	.....	.....	.....

*NOTE.—Temperatures as indicated below. Final rotor temperature after shut-down, 44° C.

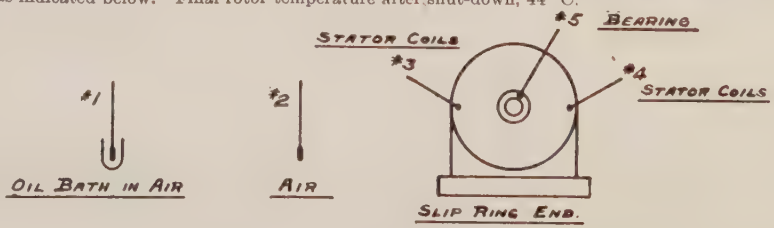


TABLE VI.—*No. 2 Pump Unit for 16th Level Station.*

Time	Motor: Volts	Amperes	Kv. A.	Kw.	Power Factor	R.P.M.	Per Cent. Slip	Pump: Head-feet: Mercury Column Suction	Ft. S.	Gauge Pressure Discharge	Ft. D	D−S+8 ft. Total	Capacity: Gauge Pressure Nozzle	Gal. per Min.	Efficiencies: Water Hp.	Motor Hp. Input	Wire to Water Efficiency	Motor Efficiency	Pump Efficiency	*Temperatures: 1	2	3	4	5
P.M.																								
12.00	2,160	68.7	257	.....	.....	.....		Hot P	acking	on	Pump—		Shut	down	.....	.....	.....	.....	.....	21.0	21.0	20.0	20.0	20.0
12.15	2,178	66.7	251	227	90.3	1,210			Noz	gle	Choked—		Shut	down	.....	.....	.....	.....	.....	21.5	21.0	23.0	23.0	25.0
1.05	2,178	100.2	377	347	92.0	1,185		.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	21.0	19.5	26.5	26.0	31.0
1.20	2,178	100.5	379	357	94.2	1,190		.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	21.0	21.0	33.0	34.0	43.0
1.35	2,178	100.5	379	350	92.4	1,190		.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	21.5	21.0	36.0	37.5	48.0
1.50	2,178	106.0	400	369	92.3	1,185		2.0	2.3	174.5	402	408	133.8	3,475	358	495	72.4	94.7	76.4	21.5	21.0	38.0	39.5	51.0
2.20	2,178	105.1	396	365	92.2			2.0	2.3	174.0	401	407	132.3	3,455	356	490	72.6	94.65	76.8	21.5	20.5	39.0	40.5	51.5
2.45	2,178	100.8	380	361	95.0			2.4	2.7	190.0	438	443	114.0	3,205	359	484	74.2	94.65	78.4	21.0	20.0	39.0	41.5	54.0
3.35	2,178	100.8	380	352	92.7			2.4	2.7	186.7	430	435	111.7	3,175	349	471	74.0	94.65	78.3	19.5	18.5	38.5	40.0	54.0
3.45	2,178	95.4	359	334	93.1			2.5	2.8	193.0	445	450	91.6	2,870	327	448	72.1	94.6	77.1	19.5	18.5	38.0	39.5	54.0
3.50	2,178	91.4	345	321	93.1			3.0	3.4	198.0	456	461	73.0	2,565	299	431	69.4	94.55	73.4	....	....	....	....	....
3.55	2,178	83.6	315	290	92.1			3.8	4.3	204.3	471	475	46.5	2,050	246	388	63.3	94.3	67.2	....	....	....	....	....
4.00	2,178	75.7	285	262	92.0			4.5	5.1	214.5	494	497	27.1	1,560	196	351	55.8	93.9	59.5	20.0	19.0	36.5	38.0	54.0
4.05	2,178	71.8	271	245	90.4			5.0	5.7	218.0	502	504	16.5	1,225	156	328	47.7	93.75	50.9	....	....	....	....	....
4.15	2,178	64.8	245	225	91.8			5.0	5.7	228.0	525	527	0.0	0	0	302	0.0	93.4	0.0	....	....	....	....	....
4.40	2,178	108.0	407	377	92.7	60 Cycles.	Generator Speed not Constant.	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	21.0	21.0	39.0	40.5	53.5

*NOTE.—Temperatures as indicated below. Final temperatures after shut-down: No. 3 42° C., No. 4 43° C., rotor 42° C. Other temperature readings shown in diagram.

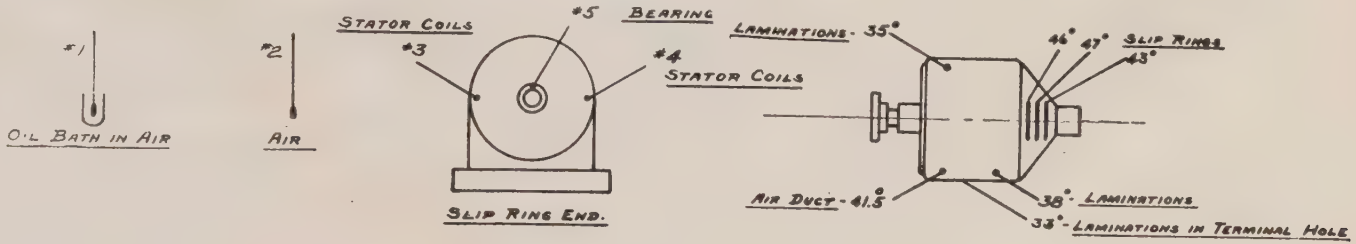
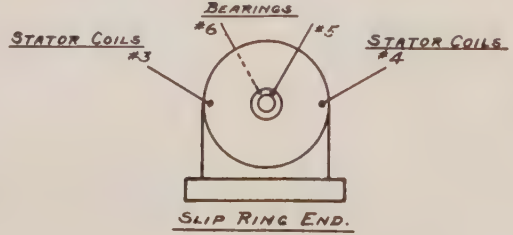
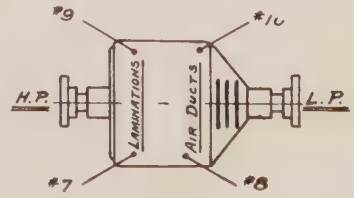


TABLE VII.—*No. 3 Pump Unit for 12th Level Station*

Time	Motor: Volts	Amperes	Kv. A.	Kw.	Power Factor	R.P.M.	Per Cent. Slip	Pump: Head-feet: Mer. Col. Suct.	Feet S	Gauge Pressure Discharge: L.P.	H.P.	Feet D	D−S+4½ ft. Total	Capacity: Gauge Pressure Nozzle	Gallons per Min.	Efficiencies: Water Hp.	Motor Hp. Input	Wire to Water Efficiency	Motor Efficiency	Pump Efficiency	Temperatures* 1	2	3	4	5	6	7	8	9	10
1.35	2,218	146.0	560	522	93.2	1,192	0.67	5.0	5.7	260.5	525	1,210	1,211	7.5	770	236	700	33.8	93.85	36.1	15.0	15.0	14.0	14.0	20.0	20.0	15.0	15.0	15.0
1.55	2,178	235.5	888	852	96.0	1,184	1.33	2.8	3.2	220.	440	1,014	1,016	103.7	3,055	784	1,141	68.6	95.05	72.2	15.0	15.0	31.0	31.0	43.0	25.0	25.0	34.0	29.0	....
2.15	2,320	244.8	984	894	90.8	1,183	1.42	2.3	2.6	187.5	384	884	886	149.5	3,672	823	1,197	68.8	95.1	72.3	15.0	15.0	43.0	42.0	49.0	43.0	32.0	41.5	38.0	41.0
2.20	2,316	243.2	975	894	91.7	1,183	1.42	2.2	2.5	188.	385	886	888	149.5	3,672	825	1,197	69.0	95.1	72.5	....	....	....	....	....	....	....	....	....	....
3.00	2,218	249.5	958	894	93.2	1,183	1.42	2.2	2.5	188.	385	886	888	149.5	3,672	825	1,197	69.0	95.1	72.5	17.0	17.0	52.0	48.0	55.0	46.0	36.5	49.0	45.0	48.0
3.05	2,218	244.8	940	874	92.9	1,182	1.50	2.5	2.8	201.	409	941	943	134.5	3,485	831	1,170	71.0	95.1	74.6	....	....	....	....	....	....	....	....	....	....
3.10	2,168	248.0	932	864	92.8	.....	....	2.5	2.8	200.	407	937	939	134.5	3,485	830	1,157	71.7	95.05	75.5	....	....	....	....	....	....	....	....	....	....
3.15	2,205	235.5	900	841	93.4	1,183	1.42	2.75	3.1	216.	440	1,014	1,016	115.0	3,220	827	1,126	73.5	95.05	77.4	13.0	8.0	53.5	50.0	56.0	47.0	38.0	49.5	39.0	48.0
3.20	2,178	235.5	888	833	93.8	.....	....	2.75	3.1	215.	440	1,014	1,016	115.0	3,220	827	1,115	74.2	95.0	78.2	14.0	12.0	51.0	47.0	55.0	46.0	36.0	47.5	40.0	47.0
3.30	2,146	230.8	858	800	93.2	1,184	1.33	3.0	3.4	227.	462	1,064	1,065	98.8	2,980	803	1,071	74.9	94.95	78.8	15.0	16.0	51.0	48.0	56.0	46.0	39.0	48.0	40.0	47.0
3.35	2,198	225.4	858	807	94.0	.....	....	3.0	3.4	229.	463	1,067	1,068	98.8	2,980	804	1,080	74.5	94.95	78.5	....	....	....	....	....	....	....	....	....	....
3.40	2,158	216.0	807	760	94.1	1,184	1.33	3.5	4.0	243.5	493	1,136	1,137	78.4	2,660	764	1,018	75.0	94.85	79.1	11.0	11.0	51.0	48.0	56.0	46.0	36.5	47.5	39.5	47.0
3.45	2,198	196.2	746	702	94.0	1,186	1.17	4.0	4.5	258.	516	1,189	1,189	56.5	2,258	679	941	72.1	94.7	76.2	....	....	....	....	....	....	....	....	....	....
3.50	2,168	199.4	749	702	93.6	.....	....	4.0	4.5	258.	517	1,191	1,191	56.5	2,258	680	941	72.1	94.7	76.2	....	....	....	....	....	....	....	....	....	....
3.55	2,178	186.8	704	659	93.5	.....	....	4.25	4.8	262.5	527	1,213	1,213	42.0	1,950	599	883	67.8	94.55	71.7	....	....	....	....	....	....	....	....	....	....
4.00	2,178	172.8	651	612	94.0	1,187	1.08	4.5	5.1	270.	543	1,251	1,251	31.4	1,680	532	820	65.0	94.35	69.0	16.5	17.0	48.0	45.0	55.5	47.0	35.0	46.0	38.0	45.0
4.05	2,188	164.8	624	574	92.0	.....	....	4.8	5.4	269.	539	1,241	1,240	26.0	1,525	480	770	62.3	94.15	66.2	12.0	12.0	46.0	45.0	58.0	50.0	33.0	45.0	39.5	45.0
4.10	2,198	117.7	448	406	90.6	1,196	0.33	5.5	6.2	268.	530	1,221	1,220	0.0	0	0	544	0.0	92.85	0.0	....	....	....	....	....	....	....	....	....	....
														0.0				0.0			....	....	45.0	44.0	....	....	42.5	49.0	43.0	49.0

* NOTE.—Temperatures as indicated below. Final temperatures after shut-down: Rotor 41° C. Slip rings—see diagram below.





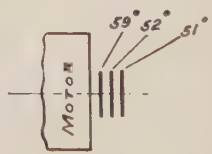
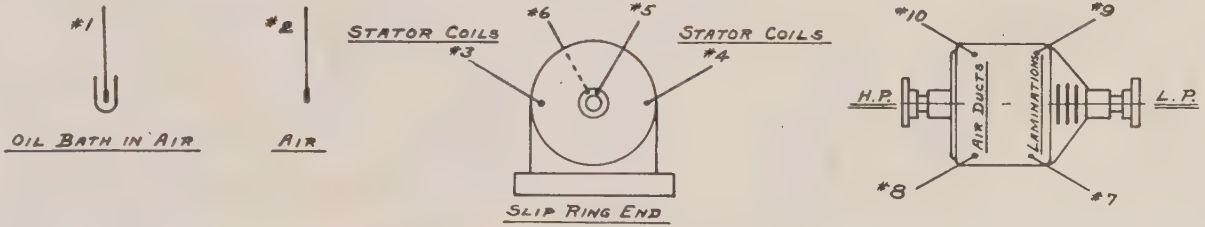


TABLE VIII.—*No. 4 Pump Unit for 12th Level Station*

Time	Motor							Pump							Efficiencies					Temperatures*									
								Head-feet						Capacity															
	Volts	Amperes	Kv.A.	Kw.	Power Factor.	R.P.M.	Per Cent. Slip	Mer. Col. Suct.	Feet S	Gauge Pressure Discharge L.P.	Gauge Pressure Discharge H.P.	Feet D	D−S+4¼ ft. Total	Gauge Pressure Nozzle	Gallons per Min.	Water Hp.	Motor Hp. Input	Wire to Water Efficiency	Motor Efficiency	Pump Efficiency	1	2	3	4	5	6	7	8	9	10
P.M.																													
9.15	2,178	141.3	533	492	92.2	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	8.0	8.0	.....	.....	.....	.....	.....	.....	.....	.....
9.30	2,178	230.8	871	815	93.5	.....	.....	.....	.....	.....	.....	.....	.....	103.7	3,055	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
9.55	2,175	250.2	944	886	93.9	1,186	1.17	1.25	1.4	177.	364	838	841	152.5	3,710	790	1,188	66.5	95.1	70.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.05	2,175	245.8	926	866	93.4	.....	.....	1.5	1.7	188.	387	891	894	139.5	3,550	804	1,162	69.2	95.1	72.8	8.0	8.0	35.0	39.0	40.0	45.0	31.0	31.0	26.0	29.0
10.10	2,178	244.8	924	869	93.9	1,186	1.17	1.5	1.7	188.	387	891	894	140.5	3,560	806	1,165	69.3	95.1	72.9	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.12	2,175	241.0	908	852	93.8	.....	.....	1.75	2.0	199.	405	932	935	128.2	3,400	805	1,142	70.5	95.1	74.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.15	2,178	233.2	880	831	94.4	1,187	1.08	2.0	2.3	209.	428	985	987	118.0	3,265	816	1,114	73.2	95.0	77.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.25	2,168	226.6	852	806	94.6	1,186	1.17	2.25	2.6	220.	448	1,032	1,034	103.7	3,055	800	1,080	74.1	94.95	78.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.30	2,178	217.5	820	772	94.1	1,188	1.00	2.5	2.8	233.	471	1,086	1,088	88.5	2,820	776	1,035	75.0	94.9	79.1	8.0	7.5	37.0	43.0	44.0	51.5	33.0	34.0	28.0	31.0
10.35	2,182	205.6	776	734	94.5	1,189	0.92	3.0	3.4	247.5	502	1,156	1,157	67.0	2,457	719	984	73.0	94.8	77.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.40	2,182	186.0	703	662	94.1	.....	.....	3.3	3.7	259.5	522	1,201	1,202	47.0	2,060	626	887	70.5	94.55	74.6	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.45	2,188	179.0	678	625	92.1	1,191	0.75	3.5	4.0	262.5	528	1,215	1,216	36.7	1,820	560	838	66.8	94.4	70.7	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.50	2,184	168.0	635	592	93.1	1,191	0.75	3.75	4.2	269.	539	1,240	1,241	27.0	1,555	488	793	65.7	94.25	69.7	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10.55	2,188	113.1	428	376	87.8	.....	.....	4.5	5.1	261.5	524	1,206	1,206	0.0	0	0	504	0.0	92.5	0.0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
11.00	2,188	250.2	948	893	94.1	1,186	1.17	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	8.0	7.5	30.0	42.0	44.0	51.0	29.0	30.0	27.0	27.0
11.30	2,188	250.2	948	893	94.1	1,186	1.17	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	8.0	7.0	39.0	43.0	47.0	53.5	33.0	29.0	35.0	30.0

* NOTE.—Temperatures as indicated below.



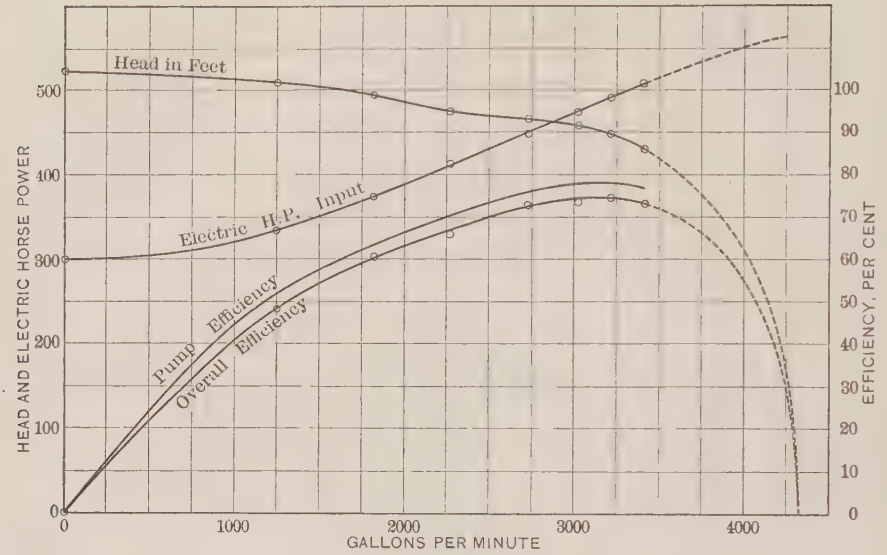


FIG. 7.—PUMP NO. 1, 16TH LEVEL.

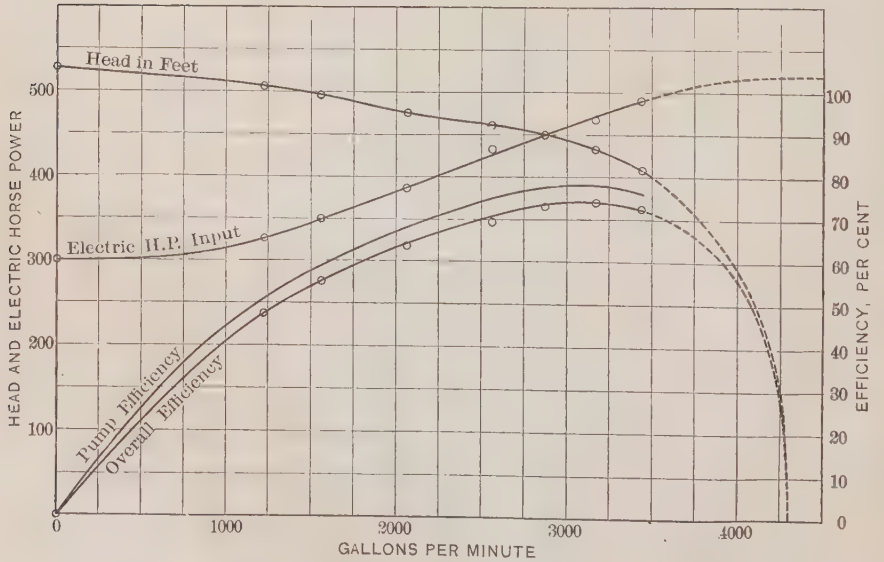


FIG. 8.—PUMP NO. 2, 16TH LEVEL.

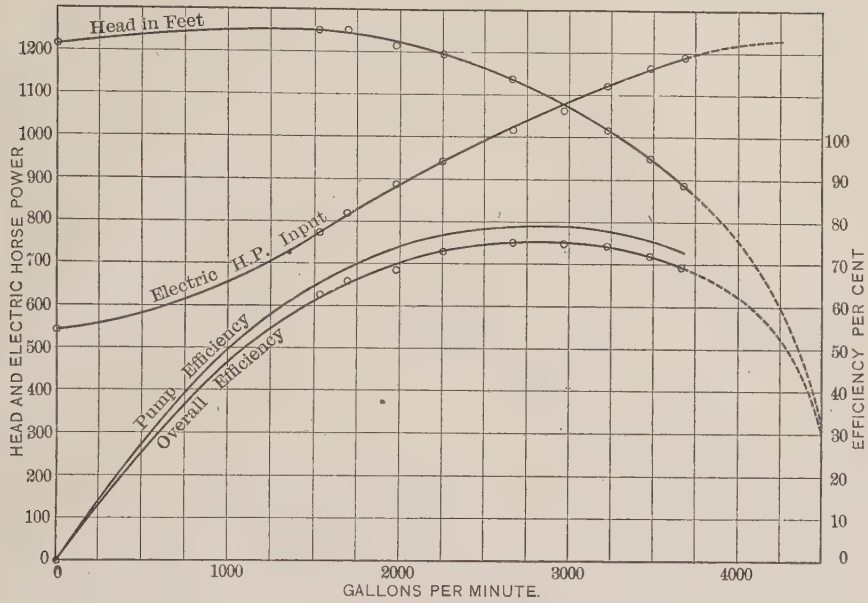


FIG. 9.—PUMP NO. 3, 12TH LEVEL.

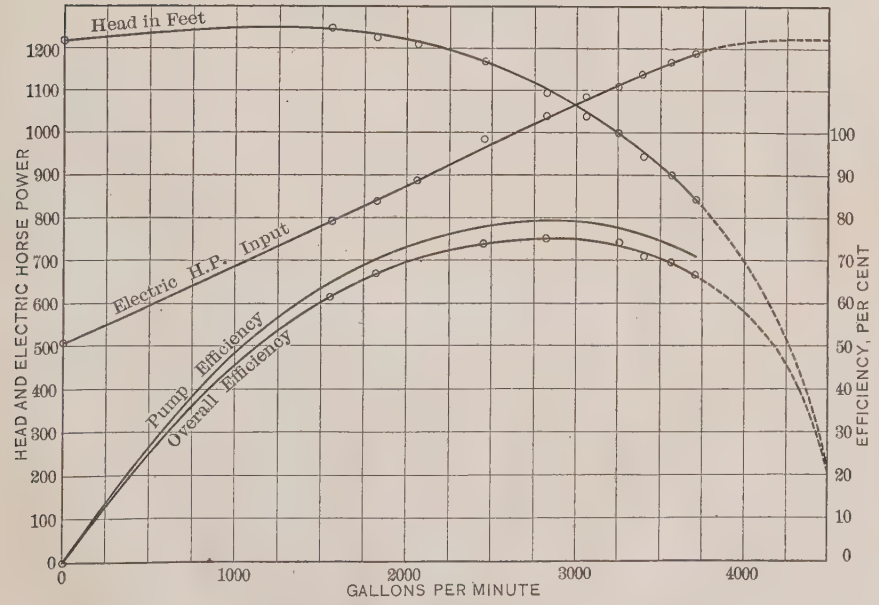


FIG. 10.—PUMP NO. 4, 12TH LEVEL.

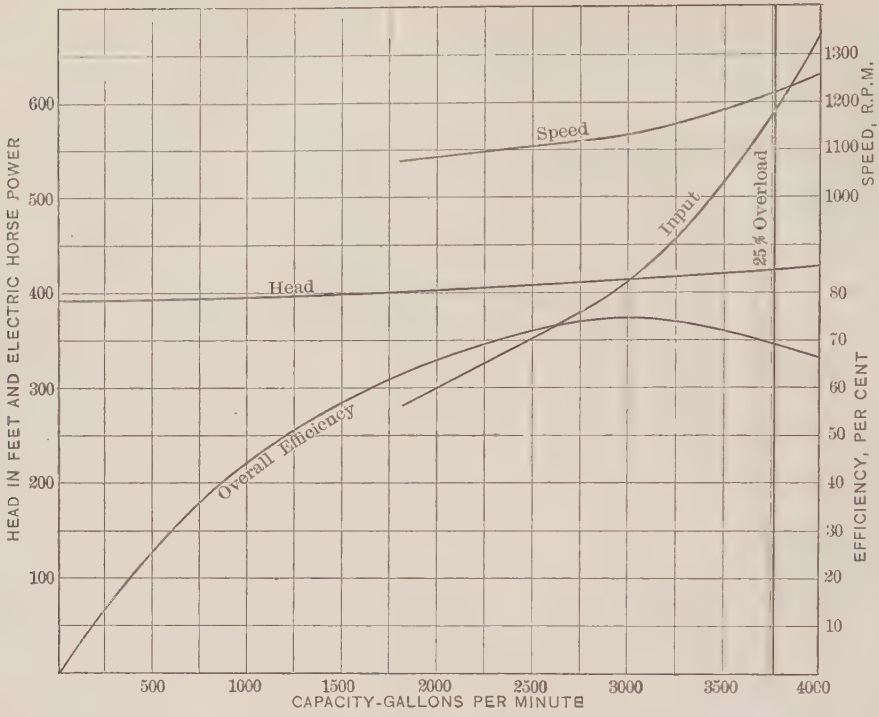


FIG. 11.—PUMP NO. 1, 16TH LEVEL.

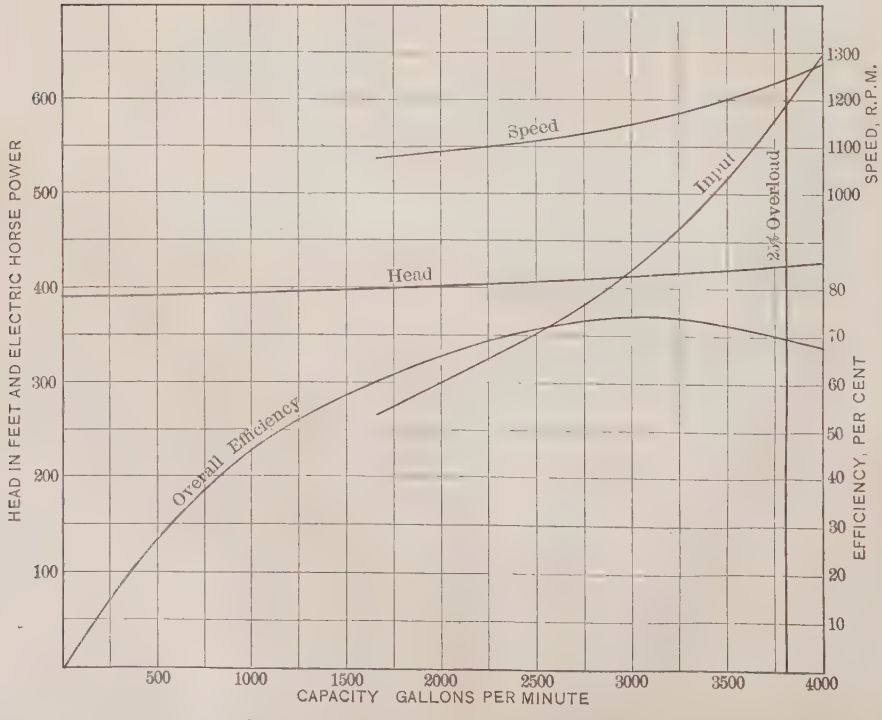


FIG. 12.—PUMP NO. 2, 16TH LEVEL.

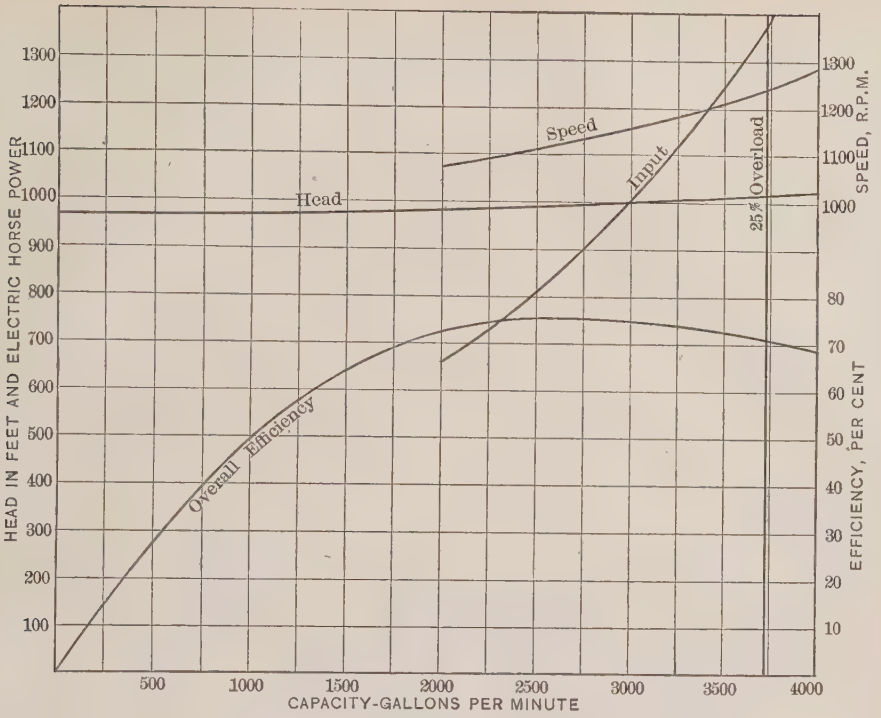


FIG. 13.—PUMP NO. 3, 12TH LEVEL.

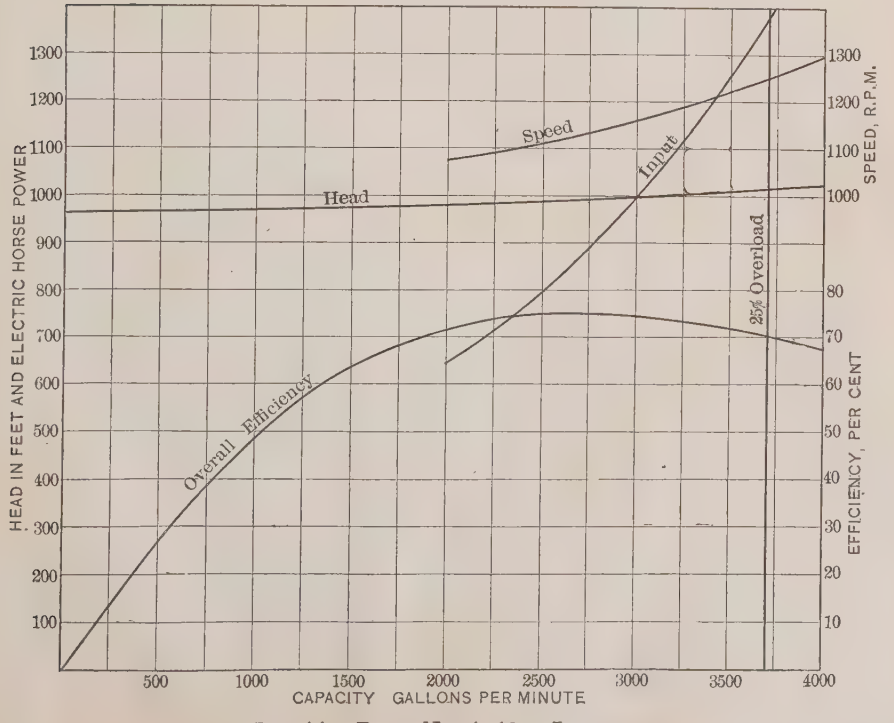


FIG. 14.—PUMP NO. 4, 12TH LEVEL.

TABLE IX

	Pump No. 1	Pump No. 2	Pump No. 3	Pump No. 4
At 3,000 gal. per minute				
Obtained head in feet	458	445	1,068	1,064
Guaranteed head in feet	412	412	1,000	1,000
Guarantee exceeded by	46	33	68	64
At 412 ft. head and 1,000 ft. head, respectively				
Obtained gallons per minute	3,550	3,415	3,275	3,250
Guaranteed gallons per minute	3,000	3,000	3,000	3,000
Guarantee exceeded by	550	415	275	250
Wire to water efficiencies at 3,000 gal.				
Obtained efficiencies, per cent	74.0	73.8	75.0	75.0
Guaranteed efficiencies, per cent	71.8	71.8	73.3	73.3
Guarantee exceeded by, per cent	2.2	2.0	1.7	1.7

TABLE X

	Time in Minutes	Temperature rise above 25° C.	Average load in Horsepower
Pump No. 1	150	25.5	457
Pump No. 2	155	20.	450
Pump No. 3	140	43.	1,200 = 14.3 per cent. overload
Pump No. 4	75	37.5	1,170 = 10.5 per cent. overload

All bearing temperatures were below normal throughout the tests. The pump bearings and oil for thrusts are water cooled and, although provision is made for the water cooling of motor bearings, no water was passed through the coils during the tests.

Some difficulty was experienced in pump thrust bearings due to foaming of oil, but this was finally overcome by a rearrangement of oil piping and oil throwers.

Mechanical Operation

On test the pumps ran very quietly at all loads and the three-stage pumps with little vibration. Some vibration occurred in the six-stage pumps, because of the difficulty in holding alignment of the long cast-

iron bases on the steel floor plates. Absolute alignment of these pumps is of prime importance and their successful operation will depend more upon this than any other one factor.

The curves in Figs. 11, 12, 13, and 14 have been prepared from the foregoing data in order to show pump characteristics at different speeds. With a variation in frequency of the prime movers from 55 to 65 cycles, a variation in pump output in gallons per minute can be obtained which will probably be within the limits required during normal pumping operations, which is at present between 3,000 and 3,250 gal. per minute. Because of the flat character of the speed-capacity curves shown, the importance of supplying a constant frequency for uniform pump output is clearly indicated.

The maximum continuous output of the pumps will be limited by the overload capacity of motors, which will probably be 25 per cent. This point is indicated on the curves.

Because of the slight difference in head-capacity curves of the three-stage and six-stage pumps, adjustment between pumps on 12th and 16th levels can be made with their discharge valves in order to pump equal quantities of water.

Discussion

K. A. Pauly, Schenectady, N. Y.—I have seen this installation and I wish all those interested in the question of electricity as a mine pumping agent could see it. It is not only a model for mine installation, but might well serve as a model for any electrically driven pumping plant. I think I am correct in saying that the pumps are the largest electrically operated centrifugal pumps in mine service in America. I talked with Mr. Rumsey in Duluth just before coming here, and he told me that the actual tests of the equipment as installed checked very closely with the shop tests; that the water delivered by the pumps was slightly in excess of the guarantee, something like 3 or 4 per cent., and that no troubles except of a very minor nature had been experienced with the installation from the start.

One point which has been brought out by Mr. Rumsey, and by others who have looked into this question, is the marked saving which can be made by the application of electricity to pumps, especially to the main sump pumps. The pumps are always located a long distance from the boiler plant and the losses in the steam mains are large. While many of the steam pumps are of a very high grade, yet the tests seemed to indicate that the steam consumption is extremely large. The reduction in operating cost of one small installation which I have in mind pumping against a head of approximately 2,000 ft.—which by the way was a reciprocating pump—reduced the pumping cost to the neighborhood of one-fifth of the previous steam cost. The high steam cost was probably

due to leakage in the engine and the loss in transmission in the mine, the pumps being located almost half a mile from the boiler plant.

From the standpoint of the manufacturers, I can say we will be very glad to hear from all of the operators as to the possibilities in electric pumping, and the conditions which should be given particular attention, especially with reference to high-head installations.

William Kelly, Vulcan, Mich.—I do not know how many here are interested in the subject of mine pumping. To those who are, it is an exceedingly important matter. The water in the mines has to have the first consideration, for if the pumps do not run nothing else is done. Where the quantity of water is considerable they are going night and day, Sundays and holidays. The selection of electrically driven pumps involves not merely a comparison between electrical pumps and steam-driven pumps, but there is a question also as to the type of electrically driven pumps—whether they should be reciprocating or centrifugal pumps. From the experience that has come to me it would seem as if each has its appropriate field. If the quantity of water is small, the reciprocating pump is doubtless the proper one to use, but when the quantity increases materially there is more of a choice and the efficiencies of the two types of pumps become more nearly equal. Efficiency as shown by a test is not always the controlling point, because tests are usually made when the machinery is working at its full capacity, but in regular pumping the supply of water is usually more or less irregular. Under such conditions there has to be some arrangement to allow a part of the water to run back, or the pump has to be stopped and started, with an adequate place for holding the water when the pump is idle. Then too comes up for consideration the difference in first cost and the interest charge, so that in each installation of mine pumping if electricity is to be used the conditions must be carefully considered and will determine the type of pump to be selected.

George S. Rice, Pittsburgh, Pa.—It has been a little difficult for me to get my bearings in the matter of the relative merits of the reciprocating pump and the centrifugal. I made my first visit to the mining districts of Europe in 1908, and I was very much astonished to find how universally the multistage electric-driven centrifugal pumps were being used at that time when they were so rarely used in mines in this country. I think in no mines in Germany or France did I find the reciprocating type. In view of the conservatism generally in England and in some districts visited it surprised me to find that that was the situation, and I had wondered why in this country our mining people, who are so generally advanced in the introduction of mechanical devices, had been—I won't say behind, but, let us say, lagging, in view of the recent developments in the introduction of the multistage pump. The latter seems to

offer such very great advantages. It is interesting to note regarding the Delaware, Lackawanna & Western mine water shaft in the vicinity of Scranton, which has automatic arrangements for lowering and reversing, the water tubs being in balance, and which I think was hoisting about 10,000 gal. a minute about 450 ft., that in subsequent enlargement of the drainage plant they put in centrifugal pumps of larger capacity, and I believe ultimately expect to supplant the hoisting plant.

In regard to the point that Mr. Kelly has brought out, that he thinks that where the quantity of water is small the reciprocating pump still holds its own: I wonder why this situation cannot be met in mining practice by running the centrifugal pump intermittently, and holding the water in sumps in the intervals, the advantage of this arrangement being conditioned on the greater efficiency of the multistage pump, at least over the old type of small mine pump.

O. P. Hood, Pittsburgh, Pa.—One of the interesting things about deep mine pumping is the inertia effect of the column of water in the discharge column. On account of the heavy pulsations, the pipe line has to be built so much stronger than is required by the static head of the water that it has added largely to the cost. Now I am asking whether in these centrifugal pumps, where the heavy pulsations are absent, the operating cost of pumping is cheapened or not.

Mr. Kelly.—I cannot answer that question exactly, but it is very evident that with centrifugal pumps there is an absence of vibration in the discharge line, which is very noticeable of course in some other pumps.

Mr. Pauly.—I can perhaps give a little information in this connection. I know that this matter has been and is being thoroughly investigated by one or more of the large valve manufacturers, one of whom told me a short time ago that he had a valve which looks very promising for eliminating a large part of the water hammer which takes place with high-head reciprocating pumps. However, any device of this kind which is subject to failure is not to be recommended.

With reference to operating pumps intermittently, this is largely a question of sump capacity and is quite common practice. I know of one company in Michigan which operates its large pumps but 8 hr. per day.

I think the electrical manufacturers are largely responsible for the slowness with which electrically driven centrifugal pumps have been adopted in this country. However, it is to a certain extent up to the mining men. In Europe the mines are worked to a point where the returns are smaller per ton and every advantage must be taken of making savings, while in this country the mines are newer and in many cases the ores are richer; the original equipment installed when the mine was opened up is still intact, and the mining men themselves have not be-

come, until recently, so intensely interested in economy as is the case in Europe. As to the manufacturer of electrical equipments, he has been doing a mail-order business, so to speak, in other lines of application. The sale of electrical equipment to the large industrial plants has been accomplished with little effort and has afforded such a wide market for purely standard apparatus that the electrical manufacturers have not until recently gone after the mining business. Further, cheap and reliable electric power has not until very recently been available in most of the camps, and the high cost of isolated plants has limited their installation.

H. H. Stoek, Urbana, Ill.—Possibly an explanation of why the first centrifugal pumps were discredited more or less is the fact that the technical press was filled by oversanguine manufacturers with prophecies that the new form would entirely replace the reciprocating. Then a few accidents and failures did a great deal to take away the credit from the new pump.

A most interesting hydro-electric plant is located near Clausthal, Germany, inside of a mine. There the water is not looked upon as a drawback, but the head is being used to drive electrical machinery. I wonder whether there are places in this country where such an arrangement would be advantageous.

Mr. Kelly.—When centrifugal pumps with high heads were first tried in this country they had not been perfected in detail. An installation at Butte was a failure on account of serious difficulties with the thrust bearings, and thus the whole system came under condemnation. Afterward that difficulty was eliminated, and centrifugal pumps were greatly improved in other particulars, but some prejudice still remains. Then, too, there is this, that in making tests at full capacity, the reciprocating pump will show a higher efficiency, but I very much question whether in a very large number of the installations they are getting in practice the efficiency which the test at full capacity shows. In practice the centrifugal pump, if in good order, will very often equal the general working efficiency of the other type of pump.

W. A. Thomas, Pittsburgh, Pa.—There are two points that have not been brought out particularly clearly, and one is the fact pertaining to Mr. Rice's question—the critical point as to head on the centrifugal type of pump. In the average coal-mining operation a pump is required which is adaptable to a wide variety of places in a mine where the heads may vary a great deal; care must be used in operating a centrifugal pump under a low head for the reason that under a half head it will tend to overload the motor seriously, and slightly over the head for which it is designed it will deliver no water at all.

Another point: in this country the reciprocating pump manufacturers

have possibly made a departure which has not been made abroad, in getting away from steam designs for electric drive. The steam-driven pump would have a crank speed of 40 to 50. Some of the manufacturers of electric pumps have decreased the size of the plungers and increased the size of the valves to get a permissible water speed through the valves and increased the revolutions of the pump to from 65 to 75, which has greatly facilitated the electric drive, because of the great reduction that had to be made from the steam speed to the motor speed. The matter of efficiency is still open for discussion and will be as long as the two types are built, which will be perpetually. Undoubtedly the quintuplex pump has the highest efficiency. The quintuplex pump manufacturers will guarantee as much as 87 per cent. efficiency. The greater number of plungers in the reciprocating type of pumps tends to reduce materially the water hammer, and that, combined with the compression chamber, has to a great extent reduced the added cost of the plunger-pump installation. Against these factors, however, is the simplicity and low cost of the turbine type of pump, which makes it popular where definite conditions of head and water are known, but it is difficult to get fractional loads, and it resolves down to the question of making a sump arrangement for gathering the water.

Mr. Kelly.—The turbine pump is a pump which is particularly suitable for a fixed head, but can be varied as to quantity. The reciprocating pump, on the other hand, can be varied greatly as to the head, but should have its full quantity of water—is not that true?

Mr. Thomas.—That is true to a certain extent; but the reciprocating pump lends itself to by-passing to get fractional loads. With the centrifugal, a slight speed difference will tend to make a great quantity difference.

R. V. Norris, Wilkes-Barre, Pa.—In the anthracite region, we have used probably every type of pump, and the centrifugal pump, we have found, is particularly applicable to conditions where the quantity of water is practically constant. With lower than the rated amount of water the efficiency of the centrifugal pump drops off rapidly, and it is necessary to run at intervals, which means supplying outfit for the maximum operations, and greatly adding to the cost. We still use a great many steam pumps in the anthracite regions.

Mr. Rice.—I would like to ask Mr. Norris a question as to the expense when the water is acid. Under those conditions is the centrifugal pump more economical to use than the reciprocating?

Mr. Norris.—I do not think it is. For very acid water I know of no instance where a centrifugal pump is used.

Edwin Ludlow, Lansford, Pa.—In the anthracite mines of the Lehigh Coal & Navigation Co., located in the Panther Valley, in the

southeastern corner of the anthracite field, we have extremely hard water conditions, as our large veins, running from 50 to 100 ft. in thickness, standing on from 45° to a vertical pitch, cause heavy surface breaks all along their outcrops on both sides of the valley, and intercept all the flood waters during heavy rains.

Our average pumping amounts to 25 tons of water to a ton of coal hoisted, and our vertical lift is from 250 to 1,000 ft.

At nearly all of our mines, we have installed water hoists, which are steam driven with 42 by 64 double-cylinder engines; they are able to hoist during the wet season a 3,600-gal. tank a minute from our 1,000-ft. levels.

At one mine, we have a drainage tunnel 4½ miles long, starting from the Lehigh River and running to our Nesquehoning shaft, intercepting it at the third level.

During a severe flood, we gauged the amount of water coming out of this drainage tunnel and found that it was flowing at the rate of 40,000 gal. a minute. This mine is located where the flood conditions are the worst. With the opening up of our mines and the letting of the water down to the deeper levels, the problem is becoming one of increasing cost each year.

The acid contained in the water has made it necessary to line the pumps and column pipes with wood, and for that reason the water hoist has been found the cheapest in annual upkeep.

We have equipped several pumps recently with cement-lined working barrels and cement-lined columns and find that "Manheim cement" withstands the acid, and has proved entirely satisfactory during the year and a half that these pumps have been in service.

We have also used a cement lining on slush pumps pumping the slush from the breakers to settling dams, and the cement has held up in that case.

On account of the heavy first cost of installation for either steam-driven or electrically driven plunger pumps, owing to the large amount of space which they require under ground, we are now experimenting with centrifugal pumps, made of all bronze and as acid resisting as possible, to be used in flood times, to assist the water hoists when the rush of water is too great for them to handle.

The largest of these pumps was built for 3,000 gal. a minute on a 750-ft. head and is 10-stage. It was not expected that these pumps would be able to maintain their efficiency in handling acid water if run continuously, but as their service will be intermittent and probably not continuous over a period of more than two months in a year, it was felt advisable to experiment with them, as the space they would require would reduce the first cost of installation to about one-half that required with plunger pumps of the same capacity.

Mr. Pauly.—I would like to ask a question with reference to the operation of centrifugal pumps operating under variable heads at constant speed. We have recently quoted on a constant-speed motor to drive a centrifugal pump, and I believe the pump has been purchased, the head on which ranges from zero to approximately 800 ft. I have been endeavoring to get a line on the pump efficiency, operated under these conditions, but have not been able to do so and would appreciate some approximate figures at varying heads.

Mr. Kelly.—It is possible to design a centrifugal pump so that the impellers can be changed. Impellers of a different diameter can be put in, which will serve to vary the capacity of the pump as to the head.

Mr. Pauly.—This is a sinking pump to which I refer and I do not think that the impellers can be changed, although I am not certain of this. The pump has been purchased in Germany, I believe, and we know little concerning it except as to the limiting dimensions to which our motor must conform.

S. S. Rumsey and W. F. Schwedes, Duluth, Minn. (communication to the Secretary*).—As the two 6-stage pumps have been in operation since July, 1, 1914, their performance in actual service may be of interest.

The pumps are installed on the 12th level, 965 ft. below the surface, as is also the Venturi meter measuring the discharge. The wattmeter measuring the input is located in the substation on surface, and it records not only the power used for pumping, but also that lost in the cable between the substation and motors, and that consumed in lighting the pump station.

In the first three columns of the accompanying table are shown the actual readings of the Venturi meter, the wattmeter, and pressure gauge in the discharge end of the pump, from which is figured the overall efficiencies in column 4. In column 5 is shown the estimated power consumed in the cable loss and pump-station lighting, which is not chargeable to the pump performance. Column 6 shows the actual overall efficiency of pump and motor as revised in accordance with the estimated losses.

Month	Actual Vent. Meter Readings Total Gal.	Total Input Kw-hr.	Average Dynamic Head	Overall Effic.	Kw-hr. in Cable and Lighting. Not Chargeable to Pumps	Net Effic., Per Cent.	Time in Operation Hr. M.	Time in Operation, Per Cent.	Average Gal. per Min.
July.....	118,326,000	523,800	991	70.25	3,480	70.70	685 5	92.1	2,876
August.....	105,470,000	460,805	991	71.15	3,400	71.68	648 30	87.2	2,708
September....	98,000,000	424,105	991	71.86	3,260	72.40	605 20	81.4	2,696
October.....	122,090,000	543,040	991	69.91	3,490	70.34	675 35	90.8	3,014
	443,886,000	1,951,750	991	70.72	13,630	71.22	2,614 28	87.8	2,830

*Received Nov. 16, 1914.

Inasmuch as the 16th-level pumps had not been placed in operation we were unable to get a continuous supply of water for the 12th-level pumps; therefore, they have operated intermittently during the four months, and on account of the insufficiency of water have been operated at about 58 cycles instead of 60.

Comparing the quantity of water indicated by the Venturi meter with the quantity estimated from the plunger displacement of the reciprocating pumps as recorded by revolution counters, with liberal deductions made for slip, we feel positive that the Venturi meter is registering between 2 and 5 per cent. less water than is actually pumped. As soon as possible we will calibrate the Venturi meter, which has not been practical up to the present time.

Refining Petroleum by Liquefied Sulphur Dioxide

BY DR. L. EDELEANU, BERLIN, GERMANY

(Pittsburgh Meeting, October, 1914)

Crude petroleum is a mixture of various groups of hydrocarbons and some bodies containing oxygen or sulphur. These constituents possess properties differing considerably one from another and the proportion in which they are contained in the crude oils is also very different.

Owing to this complicated composition the practical use of crude petroleum is much restricted.

For example, it is sometimes used as a liquid fuel. But its employment in this way is uneconomical because various valuable qualities of the individual constituents are not made use of.

The rational utilization of petroleum can only be accomplished if the various groups of hydrocarbons are separated. The mixture of hydrocarbons present in the crude oil must be split up in such a way that the hydrocarbons are divided into various groups having certain predominant characteristics and uses. To accomplish such a separation is the aim of the technical refining of petroleum.

Ordinary Refining by Distillation and Chemical Treatment

Hitherto this has been accomplished by two processes, namely, fractional distillation, and chemical treatment of the distillates obtained.

Distillation Process

The process of distillation consists in the separation of the constituents of petroleum into various fractions, the boiling points of which lie within certain defined limits. The fractions so separated possess certain special qualities in common, which enable them to be applied for specific purposes in practice. For example, we have the fractions known as benzene, illuminating oils, heavy oils, lubricating oils, etc.

The benzene fraction, which contains the readily volatile hydrocarbons, furnishes chiefly the well-known solvents, and is now mainly used for producing the driving power for automobiles.

The intermediate fractions boiling up to about 300° C. are the oils used for purposes of illumination. This product must comply with several requirements. It should not smoke, should not possess any unpleasant odor, and should give a bright and steady flame which does not

undergo unduly great alterations while burning. Now these conditions are only complied with by oils of certain origin, while many other oils do not fulfill the requirements in the least; it is therefore one of the special problems in refining to convert these inferior varieties into good burning oils.

The heavier fractions are used as gas oil or material for Diesel motors and the more viscous fractions are employed as lubricants.

For use as lubricants these fractions must possess a viscosity which should be retained within certain limits of temperature variation, such as are commonly met with in the practical use. Moreover, there must be no deposit of free carbon and they must be freed from components which at high temperature would form a deposit of char and thus diminish the value of the lubrication.

Now the object of the petroleum refinery cannot be achieved by distillation alone. In fractional distillation it is inevitable that some varieties of hydrocarbons having the same boiling point will find their way into certain fractions, notwithstanding the fact that their chemical properties are distinctly dissimilar. Distillation alone is thus quite unsuitable. The problem to be solved is how to retain only the valuable hydrocarbons, and to eliminate those with undesirable or injurious properties.

Refining with Sulphuric Acid

Consequently recourse has been had to the chemical treatment of petroleum. In this treatment sulphuric acid is mainly used for the purpose of removing the resinous matters and certain hydrocarbons such as olefines. These constituents are separated out in the form of residual sludge—impurities which can be utilized in practice only with extreme difficulty or possibly not at all. For the removal of the aromatic hydrocarbons considerable quantities of sulphuric acid, sometimes in the state of "fuming acid," have to be used according to the proportion of these hydrocarbons present in the distillate and this renders the process distinctly costly. The application of large quantities of sulphuric acid and especially in the state of fuming has a further drawback. It does not attack only the aromatic and unsaturated hydrocarbons, but also destroys a certain quantity of the saturated constituents. The process has thus the disadvantage of still further reducing the proportion of useful hydrocarbons finally obtained.

In order to avoid the drawbacks of the treatment hitherto employed, it is necessary to abstain from the purely chemical treatment and to substitute a physical method of extraction. The constituents of the petroleum must be treated with a solvent which dissolves the hydrocarbons of the unsaturated aromatic group but leaves the saturated hydrocarbons undissolved. In this way the two classes of hydrocarbons can be separated from one another and employed in the way for which they are suited.

The most important point in the solution of this problem consisted in the discovery of an appropriate solvent, and finally liquefied sulphur dioxide was found to be the best material. This substance readily dissolves the unsaturated hydrocarbons, which are responsible for the unsatisfactory burning and odor characteristic of certain illuminating oils, and leaves undissolved the useful illuminating oil. Frasch attempted to achieve the same object by the application of alcohol, but liquefied sulphur dioxide has certain advantages which alcohol has not. The chief advantage, apart from cheapness, consists in the fact that liquefied sulphur dioxide can be readily recovered from the solution, so that in practice the solvent can be used again and again in a cycle of operations, without any appreciable loss.

New Refining Process with Liquefied Sulphur Dioxide

The new process makes use of this peculiar property of liquid sulphur dioxide. If the distillate is agitated with liquid sulphur dioxide at a low temperature the aromatic compounds are dissolved, but the paraffins and naphthalenes are unaffected. Moreover, owing to the difference in the specific gravity, two distinct layers are formed so that it is quite easy to separate them from one another.

The process is comparatively simple. A certain quantity of the sulphur dioxide is added to the petroleum distillate and is dissolved. This addition is continued until the distillate is saturated, and, from that point, each added portion of sulphur dioxide dissolves the aromatic constituents and separates out with them as a distinct layer. Thus, as the treatment is continued, the distillate becomes poorer and poorer in aromatic hydrocarbons until ultimately a refined distillate is obtained, which contains only a small amount of sulphur dioxide, and is practically free from the objectionable aromatic constituents.

It may be observed that the nature of this operation depends to some extent on the temperature. With a rising temperature the solubility of the paraffin hydrocarbons and the naphthalenes—which at low temperatures are practically insoluble—increases rapidly; whereas the aromatic hydrocarbons are soluble in sulphur dioxide at *all* temperatures. It is therefore important that the temperature should be maintained at a suitably low level, particularly in the case of distillate rich in aromatic hydrocarbons. The degree to which the process is affected by the temperature naturally depends on the nature of the material treated. For instance, in the case of a distillate from Borneo, which contained as much as 40 per cent. of aromatic hydrocarbons, treatment with 66 per cent. liquid sulphur dioxide at $-7°$ C. produced no separation; but a considerable proportion of the aromatic constituents could be separated at $-10°$. On the other hand, in the case of a Mexican distillate, which contained only about 17 per cent. of aromatic hydrocar-

bons, treatment with an equal amount of liquid sulphur dioxide produced separation even at a temperature higher than +10° C. In practice it is found that each distillate requires special treatment, according to its composition, in order to obtain the best results.

This process, which is now being used on a large scale in the petroleum industry, has been worked out by the author also as a laboratory method. For the experiments he uses:

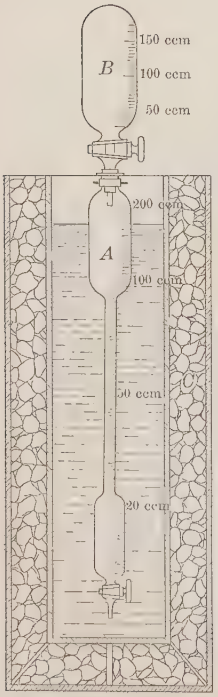


Fig. 1

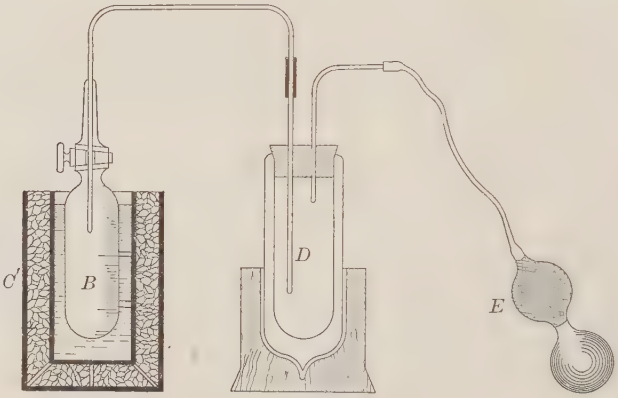


Fig. 2

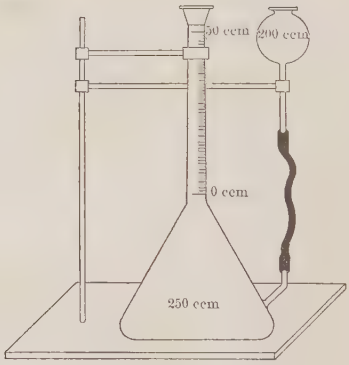


Fig. 3

1. A strong-walled graduated burette of 200 cm. contents drawn out pear-shape like on each end with a delivery cock protected against being forced out. (See Fig. 1, *A*.)

2. A strong-walled cylindrical graduated vessel of about 230 cm. contents with a delivery cock also protected against being forced out. (See Fig. 1, *B*.)

3. Two cooling baths for vessels 1 and 2. The cooling bath is insulated on the outside by means of cork sheets and consists of two tin cylinders which telescope into each other concentrically. The outer

cylinder contains a freezing mixture, and the inner a liquid which conducts cold, preferably a distillate of illuminating oil. (See Figs. 1 and 2, *C* and *C'*.)

4. A flask with a graduated neck and a funnel attached to it. (See Fig. 3.)

5. A sprinkling flask for liquefied sulphur dioxide, consisting of an evacuated double-walled Dewar vessel. (See Fig. 2, *D* and *E*.)

To carry out the experiment, one proceeds as follows:

The graduated burette is filled with 50 cm. of the distillate to be examined, and the cylindrical vessel with double the quantity of the distillate of liquefied sulphur dioxide. Both liquids are cooled in the cooling bath to $-12°$ C. The filling of the cylindrical vessel with sulphur dioxide is illustrated in Fig. 2. Then the cylindrical vessel is connected, by means of a tight-closing cork stopper, with the burette, and the connection is reinforced either by copper wire or by a suitable clamp. Then sulphur dioxide is allowed to flow from the cylindrical vessel into the burette until in the lower part of the burette, after shaking, a permanently remaining layer is observed. Then one-third the quantity of the balance of sulphur dioxide is allowed to run into the burette. It is again cooled to $-12°$ C., shaken, and then the lower layer is drawn off into a Dewar vessel. In the same manner, the remaining portion of the sulphur dioxide is introduced, and the sulphur dioxide solution is subsequently allowed to run into the same vessel. Finally, the non-dissolved saturated hydrocarbons remain in the burette.

In treating high-boiling distillates, such as gas oil or lubricating oil, it is sufficient to allow the sulphur dioxide to escape from the two separated products and to establish the ratio of the two groups of hydrocarbons, either by direct weighing or by specific gravity.

In treating relatively low-boiling distillates, such as benzene and illuminating oil, losses may be suffered if the sulphur dioxide is allowed to evaporate voluntarily. In order to avoid these losses it is preferable not to evaporate the sulphurous acid, but to absorb it in a closed vessel by means of an alkaline solution. In this case, one proceeds as follows:

After the sulphur dioxide solution has been drawn off, the burette is connected, by means of a cork stopper, with a glass vessel, and gradually under stirring the oil layer treated with sulphurous acid is allowed to run into an alkaline solution. After all the sulphurous acid has been neutralized and the contents of the vessel have been cooled, the stopper is opened and water is allowed to run in until the whole of the oil has cooled in the graduated part of the neck. Then the volume of the oil layer is determined; also the specific gravity, and from the data obtained the quantity of saturated hydrocarbon is established. The difference between this weight and the weight of the original distillate gives the amount of aromatic and cyclic unsaturated hydrocarbons.

Table I gives the results of a very large number of experiments with illuminating oil of various origins. Further experiments to extend this method also for the examination of crude oil are in the course of being worked out.

TABLE I.—*Experiments with Illuminating-Oil Distillates*

A = Specific gravity before treatment with SO_2; B = specific gravity after treatment with SO_2; C = paraffin- and naphthene-hydrocarbons in per cent. by weight; D = aromatic and cyclic unsaturated hydrocarbons in per cent. by weight.

Origin of Petroleum	A	B	C	D
EUROPE:				
Roumania:				
Bustenari	0.8195	0.8030	75.0	25.0
Câmpina	0.8125	0.7968	82.9	17.1
Moreni	0.8185	0.8050	77.0	23.0
Tintea	0.8195	0.8090	82.5	17.5
Filipesti	0.8104	0.7984	84.7	15.3
Policiori	0.8073	0.7955	84.3	15.7
Galicia:				
Urycz	0.8230	0.8105	80.6	19.4
Rogi	0.8120	0.8010	81.3	18.7
Tustanovice	0.8105	0.8000	80.8	19.2
Boryslaw	0.8166	0.8028	83.1	16.9
Germany:				
Wietze	0.8200	0.8098	87.4	12.6
Pechelbronn	0.8058	0.7992	92.3	7.7
Scotland	0.800	0.7815	78.3	21.7
Scotland	0.7915	0.7807	80.4	19.6
Italy	0.8030	0.7902	64.7	35.3
ASIA:				
Russia:				
Bibi-Eybat (Baku)	0.8160	0.8085	92.7	7.3
Bibi-Eybat (Baku)	0.8145	0.8092	91.7	8.3
Balachany (Baku)	0.8192	0.8150	93.1	6.9
Grossny	0.8170	0.8072	85.6	4.4
Grossny	0.8155	0.8062	83.4	16.6
Maikopp	0.8165	0.800	71.5	28.5
British India:				
Burmah	0.814	0.799	80.7	19.3
Burmah	0.817	0.800	80.1	19.9
Burmah	0.840	0.8265	83.2	16.8
Persia	0.7988	0.7862	83.2	16.8
Holl. India:				
Borneo	0.8480	0.8035	58.6	43.4
Borneo	0.8505	0.8090	55.5	44.5
Sumatra	0.7994	0.7865	83.8	16.2
AMERICA:				
United States:				
Pennsylvania	0.7960	0.7920	93.5	6.5
Ohio	0.7990	0.7926	95.1	4.9
Ohio	0.7985	0.7892	90.9	9.1

Texas	0.8240	0.816	87.5	12.5
California	0.8098	0.7982	85.0	15.0
California	0.8100	0.803	90.5	9.5
California	0.8124	0.8015	81.8	18.2
California	0.8190	0.8112	80.8	19.2
California	0.8195	0.8084	81.4	18.6
California	0.8194	0.8112	85.6	14.4
Mexico	0.8020	0.7926	84.0	16.0
Mexico	0.8030	0.7930	83.9	16.1
Mexico	0.8082	0.7972	82.9	13.1
Peru:				
Lobitos	0.8174	0.807	81.8	18.2
Lobitos	0.8175	0.8122	89.5	10.5
Argentina	0.800	0.793	90.7	9.3
Argentina	0.810	0.804	90.4	9.6
AFRICA:				
Egypt	0.8104	0.7976	82.5	17.5

TABLE II.—*Comparison of the Qualities of Illuminating Oils Obtained by the Ordinary and by the Edeleanu Process*

Origin	Refining Process	Specific Gravity	Illuminating Power: Photometric Measurements at each Hour with the Kosmos Burner 14. Illuminating Power in Hefner Units								
			0	1	2	3	4	5	6	7	8
Bustenari	Ordinary process	0.818	7.2	7.1	7.1	7.2	7.1	7.1	7.1	6.7	6.7
	SO_2 process	0.803	13.8	13.3	13.2	13.2	13.2	13.0	12.8	12.5	12.4
Câmpina	Ordinary process	0.812	10.4	10.1	10.1	9.9	9.9	9.9	9.9	9.9	9.9
	SO_2 process	0.797	15.7	15.5	15.3	15.3	15.3	14.9	14.3	14.3	13.5
Moreni	Ordinary process	0.818	7.3	6.7	6.4	6.3	5.6	5.6	5.6	5.3	5.1
	SO_2 process	0.805	13.5	12.8	11.8	11.2	11.2	11.2	10.9	10.8	10.8
Tustanovice	Ordinary process	0.799	13.8	13.7	13.1	13.1	12.9	12.9	12.4	12.0	11.7
	SO_2 process	0.786	17.2	17.2	16.7	16.0	15.5	15.0	14.4	14.0	13.7
Grossny	Ordinary process	0.815	10.0	10.0	9.9	9.8	9.8	9.6	9.2	9.1	8.9
	SO_2 process	0.806	15.4	15.0	14.7	14.3	13.9	12.8	12.4	12.1	11.8
California	Ordinary process	0.810	9.4	9.1	8.9	8.6	8.4	8.2	8.1	8.1	8.1
	SO_2 process	0.798	14.7	14.3	13.9	13.5	13.2	13.0	12.8	12.8	12.6
Peru	Ordinary process	0.817	11.2	10.7	10.4	9.9	9.6	9.4	9.2	8.9	8.9
	SO_2 process	0.807	14.3	13.8	13.6	13.6	13.5	13.5	13.2	12.6	12.1
Qualities of Illuminating Oil Brands Sold on the Market											
Pennsylvania W. W		0.7915	14.3	14.1	13.9	13.5	12.9	12.8	11.6	10.1	9.9
Pennsylvania St. W		0.7995	12.8	12.2	11.9	11.8	11.2	11.0	10.2	9.7	9.2
Bakou Meteor		0.8087	11.1	10.8	10.8	10.6	10.6	10.6	10.2	10.0	9.9

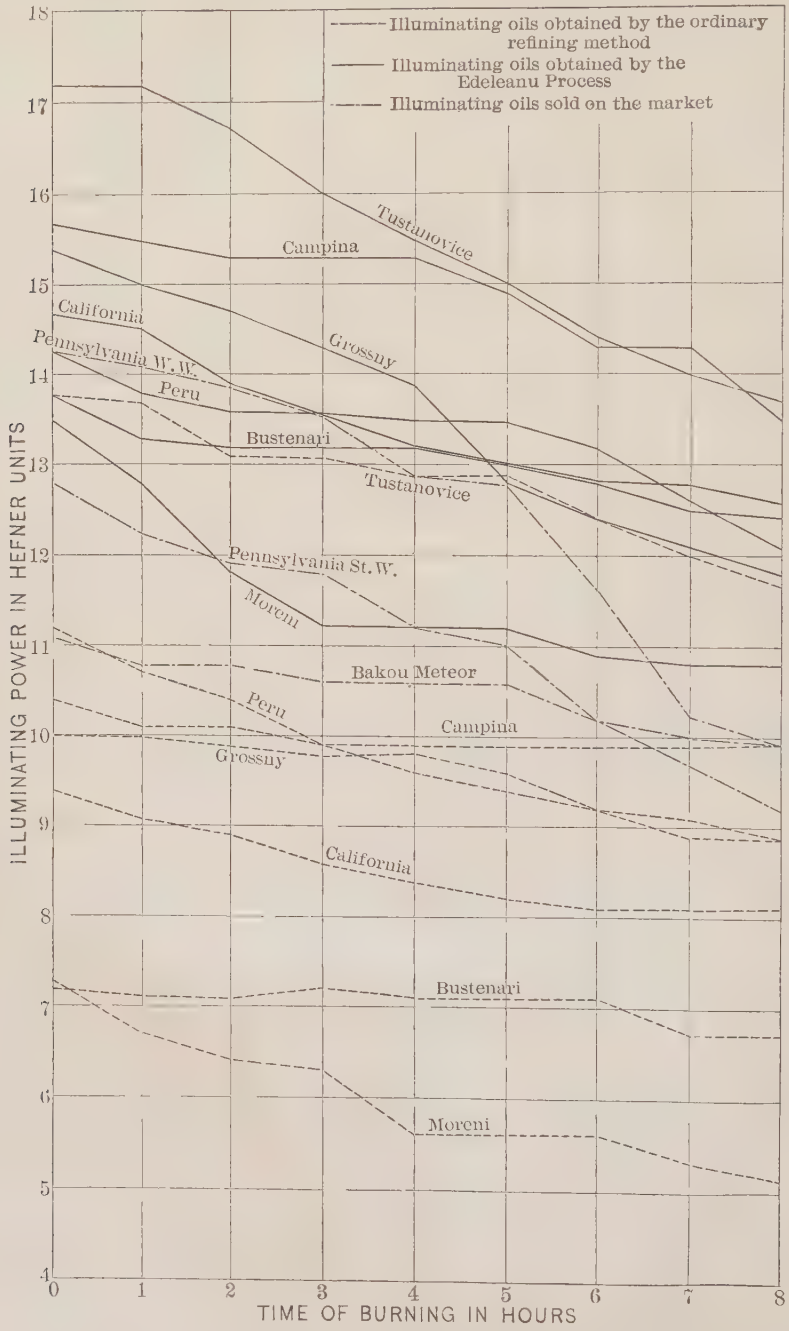


FIG. 4.—COMPARISON OF OILS FROM ORDINARY AND EDELEANU PROCESSES.

The removal of the aromatic hydrocarbons from an illuminating-oil distillate causes not only a lowering of the specific gravity, but also the raising of the burning capacity and illuminating power. The richer a distillate is in aromatic and heavy hydrocarbons, the more distinctly stands out the improvement brought about by treatment with sulphurous acid. The effect of the new process is clearly discerned when treating a Bustenari distillate. Whereas the chemical process hitherto employed produces a very inferior illuminating oil which smokes badly, and when burned in a 14-unit Cosmos burner develops only 7 cp., the treatment with liquid sulphur dioxide produces an oil equal in quality to the best American and Russian oils.

Table II illustrates the action of the old and new methods respectively on distillates of various origin. See also the graphical comparison in Fig. 4.

Utilization of By-Products

The extract obtained in addition to illuminating oil contains, as mentioned, aromatic hydrocarbons and hydrocarbons rich in carbon. On account of their low burning capacity, they cannot be employed as burning oils; nevertheless, by dividing the extract into two products by redistillation, they become useful; the lighter part as turpentine substitutes or mixed with light benzene as a motor spirit of high efficiency, and the heavier as motor oil for Diesel motors. The extract is, therefore, as regards the possibility of turning it to account, at least equally as valuable as the distillate.

TABLE III.—*Comparison of Tar Extract and Coal Tar*

No.	Composition of Tar Extract				Average Composition of Coal Tar according to Prof. Kramer	
	Limits of Temperature ° Celsius	Density at 15° C	Products of Distillation	Per Cent. Calculated for Tar	Products of Distillation	Per Cent.
1	80°—113°	0.8700	Benzene, toluene	13.3	Benzene et Homol	2.50
2	113°—140°	0.8700	Little toluene	.....	Phenols.....	2.00
			with much xylene	9.1	Residue.....	0.25
3	140°—250°	0.9870	*a.* Naphthalene	8.7	Naphthalene	.6.00
			b. Hydrocarb.		Heavy oil.....	20.00
			arom. liquids...	23.7	Anthracene, Phenantrene.....	2.00
4	250°—400°	.....	*a.* Anthracene crude.....	1.0	Asphalt.....	38.00
			b. Anthracene oil.....	20.0		
5	above 400°	.....	Residue.....	21.2	Coke.....	24.00
			Loss.....	3.0	Water.....	4.00

Further experiments have shown that the extract can be resolved by special means into the lower homologues of the aromatic hydrocarbons. By pyrogenic distillation the hydrocarbons contained in the extract are split up; and, besides a gaseous part, a large percentage of tar is produced. The latter is distinguished from tar derived from coal by its higher content of the lower members of the aromatic series of hydrocarbons.

Table III shows the products of gasification (pyrogenic distillation) of the extract as compared with other bituminous substances.

While coal tar hardly contains 2.6 per cent. of aromatic hydrocarbons, the tar produced by pyrogenic distillation of petroleum contains between 15 to 23 per cent.

Among the fractions of this tar which distill above 200° a large quantity of naphthalene is found, which varies between 5 and 12 per cent., and of anthracene varying between 0.5 and 2.7 per cent.

The benzene which one obtains after the first redistillation of this tar is much purer than that which one obtains under the same conditions by redistilling coal tar. This result appears from Table IV, which shows comparatively the composition of benzene obtained by the first redistillation of tar from the extract (treatment with dioxide of sulphur), and that of the type of benzene used in commerce.

TABLE IV.—*Composition of Commercial Brands of Benzene*

Commercial Benzene				Benzene from Extract 80°—113° C.	
Limits of Temperature, ° Celsius	30 Per Cent.	50 Per Cent.	90 Per Cent.	Limits of Temperature ° Celsius	Per Cent.
80°	0	0	25	85°	0.0
90°	2	4	70	90°	13.4
95°	12	26	83	95°	48.0
100°	30	50	90	100°	74.0
105°	42	62	94	105°	88.0
110°	70	71	97	110°	96.0
115°	82	82	98	115°	98.0
120°	90	90	99	120°	99.0

As it appears from this table, the fraction of 80° to 103° which one obtains by the pyrogenic distillation of the petroleum extract corresponds to an intermediate quality which ranges between the 50 per cent. type and the 90 per cent. type, as sold in the market, and such as is obtained from coal tar by means of several redistillations. It must also be remembered that the benzene obtained by the pyrogenic treatment of the extract and of other products of petroleum is totally free from thiophen

and contains few impurities. By means of a little soda it is obtained perfectly pure.

Effect of Process on Distillates Containing Sulphur

It is further to be noted that the liquid sulphur dioxide also possesses the property of dissolving out from crude distillates certain sulphur-containing constituents. For instance, by treating a Mexican distillate of 0.803 specific gravity and with a sulphur content of 0.6 per cent. an oil is obtained which shows a percentage of sulphur of only 0.08 per cent. and another oil of 0.79 specific gravity, with a sulphur content of 0.46, could be reduced to a percentage of sulphur of 0.04.

The process was first tested in a small experimental plant in the Vega refinery at Ploesti.

The results obtained with this small experimental plant were entirely satisfactory and shortly afterward the construction of the necessary plant for carrying out the process on a large scale was undertaken by the firm of A. Borsig, of Tegel (Berlin). A refining plant was next installed in Rouen, and was followed soon after by the construction of a plant of similar size in Ploesti. The capacity of the plant at Ploesti, originally intended to treat up to 25 tons of distillate per day, was soon increased to 70 tons a day and the products are giving every satisfaction.

These trials were considered to demonstrate completely the practicability of the process on a large scale. One of the largest and most influential petroleum concerns has since ordered a plant with a daily capacity of 250 tons, and already an increase to 500 tons is being contemplated. There are now other large plants in course of construction for Galicia, Roumania, Borneo, India, etc.

The following is a description of the treatment of a Roumanian distillate from Bustenari:

The dried distillate to be treated passes through filters charged with salt, is collected in a receiver, and finally reaches the distillation cooler after going through a cold-exchanging apparatus. The temperature of the distillate is first lowered in the cold-exchanging apparatus, and it is then further cooled in the distillation cooler itself until it reaches the desired low temperature. Similarly, the liquid sulphur dioxide coming from the vessel passes through a second cold-exchanging apparatus, where its temperature is reduced somewhat, and from thence to the sulphur dioxide cooler itself, where it receives the necessary final cooling. The arrangement is shown in Fig. 5.

The distillate to about $-10°$ C. is now let into the lower mixer, a cylindrical vessel which is provided with level gauges, and in its lower part with gauge glasses, so that the reaction in the liquid may be accurately observed.

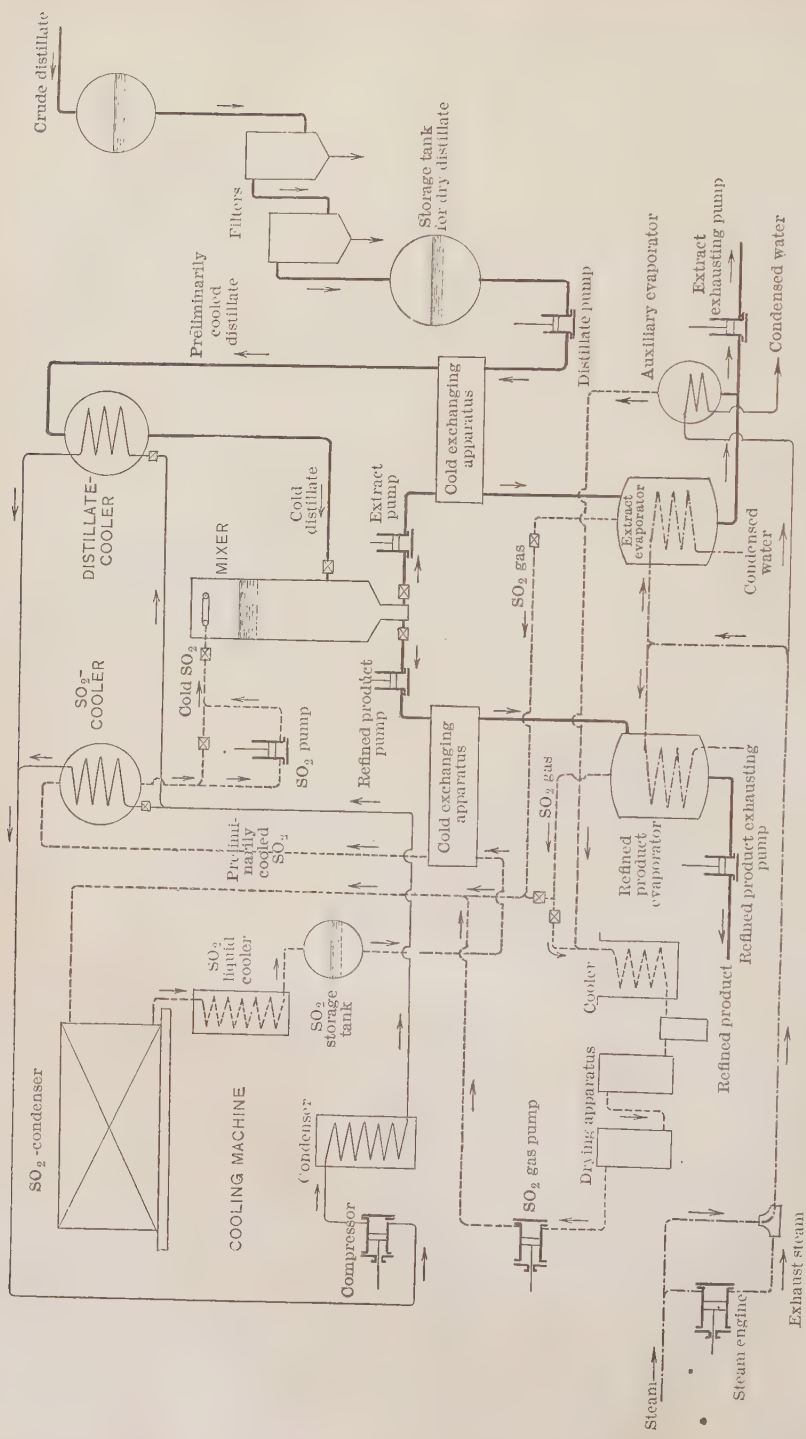


FIG. 5.—PLAN OF A REFINING PLANT FOR LIQUEFIED SULPHUR DIOXIDE PROCESS.

The introduction of the liquid sulphur dioxide into the mixer begins immediately. The sulphur dioxide is at first entirely absorbed by the distillate, without any appreciable change of the liquid. As soon as the distillate is saturated with the liquid sulphur dioxide, dark cloud masses begin to rise, and the color changes suddenly to a deep brown. This change of color is brought about by the separating out of the liquid sulphur dioxide loaded with the aromatic hydrocarbons, and depositing at the bottom of the mixer. There are now two sharply divided layers which can be distinctly observed through the level gauges. As soon as there is a certain quantity of extract formed, the drawing-off commences; at the same time further quantities of liquid sulphur dioxide are supplied to the mixer.

The extract is drawn by the extract pump through the cold-exchanging apparatus into the extract evaporator. In the cold-exchanging apparatus it meets the comparatively warm distillate flowing to the distillation cooler (as described above), by which its temperature is considerably raised. In this way the distillate receives a preliminary cooling and the extract a preliminary warming. The latter is then further warmed in the extract evaporator, which is provided with heating coils. This causes the rapid evaporation of the dissolved sulphur dioxide which flows directly into a condenser, where it accumulates in the form of a liquid; it is then stored in the reservoir, from whence it commences its cycle afresh.

As soon as the greater part of the sulphur dioxide contained in the extract has been driven off, the evaporation is gradually brought to a stop. The final remaining portion of the sulphur dioxide is held very tenaciously by the extract, and cannot be driven off merely by the application of heat. In order to prepare the extract evaporator for a fresh charge, the extract with its still remaining small content of sulphur dioxide is run into an auxiliary evaporator. Here, the balance of the sulphur dioxide is drawn off by a sulphur dioxide gas pump under continuous application of heat, and sent to the condenser. This is continued until about 0.3 per cent. still remains in the extract, when the latter is removed from the auxiliary evaporator.

The refined product is pumped through the second cold-exchanging apparatus into the second evaporator. It meets in the cold-exchanging apparatus the liquid sulphur dioxide flowing from the tank to the cooler, and is raised to practically the same temperature. In this way the liquid sulphur dioxide receives a preliminary cooling (just as in the case of the distillate). The refined product is then collected in the evaporator and undergoes a similar process of evaporation as the extract. By means of heat and continued suction the sulphur dioxide is almost completely removed, only about 0.1 or 0.2 per cent. remaining; at which stage the refined-product evaporator is emptied.

The treated distillate, after being washed and neutralized with an

alkaline solution, may be used as finished illuminating oil. In some cases subsequent treatment with very small quantities of sulphur dioxide is advantageous in order to obtain a "water-white" color.

The exhaust steam from the steam engine which drives the plant is utilized for the heating of the evaporators.

While the evaporating process is still going on in the extract evaporator a new operation takes place, a freshly cooled distillate being admitted to the mixer and saturated with liquid sulphur dioxide, until again fresh quantities of extract settle out. By this time the evaporation in the extract evaporator is also finished and the apparatus emptied, so that the new operation is able to proceed without interruption.

The cooling of the distillate and sulphur dioxide collected in the two cooling vessels is effected in the usual manner with the help of cooling coils, which form a part of a separately working cooling machine. The cold-producing medium enters the cooling tubes as a liquid, evaporates on account of the continuous low pressure produced by the suction of the compressor of the cooling machine, and abstracts from its surroundings, *i. e.*, the distillate and liquid sulphur dioxide, the heat required for its evaporation. The vapor of this cooling medium is afterward sucked up by the compressor and led to the condenser of the cooling machine, where it is again collected in liquid form.

The method of working is shown graphically in Fig. 6. In this diagram it is shown how fresh distillate is continuously entering the system, and how it finally leaves the system split up into refined product and extract, and how the sulphur dioxide, which effects the splitting up, is recovered with the exception of a very small quantity, and is then used again for the process, making a complete cycle. It may also be seen that of the distillate put into the system nothing is lost; the products treated, viz., the volume of extract and the refined product, together are equal in volume to that of the original distillates.

As the sulphur dioxide performs a complete cycle, it cannot, when once entering the system in a pure and anhydrous condition, give rise to any undesirable complications. It is well known that liquid sulphur dioxide is quite inactive toward metals, so that there is no fear of the apparatus becoming injured or destroyed in the course of time.

However, great care must be taken that no moisture is introduced into the system with the distillate. The latter should, therefore, as mentioned above, pass through several drying filters before it enters the apparatus, so that all the moisture may be removed. Inasmuch as the system is perfectly airtight throughout, there is of course no possibility of moisture from the atmosphere penetrating into it.

The fact that the whole of the apparatus, pipes, valves, etc., are completely airtight is responsible for the fact that scarcely the faintest smell of sulphur dioxide is perceived in the plant. This is of the greatest

importance for the welfare of the workmen who are in attendance at the plant.

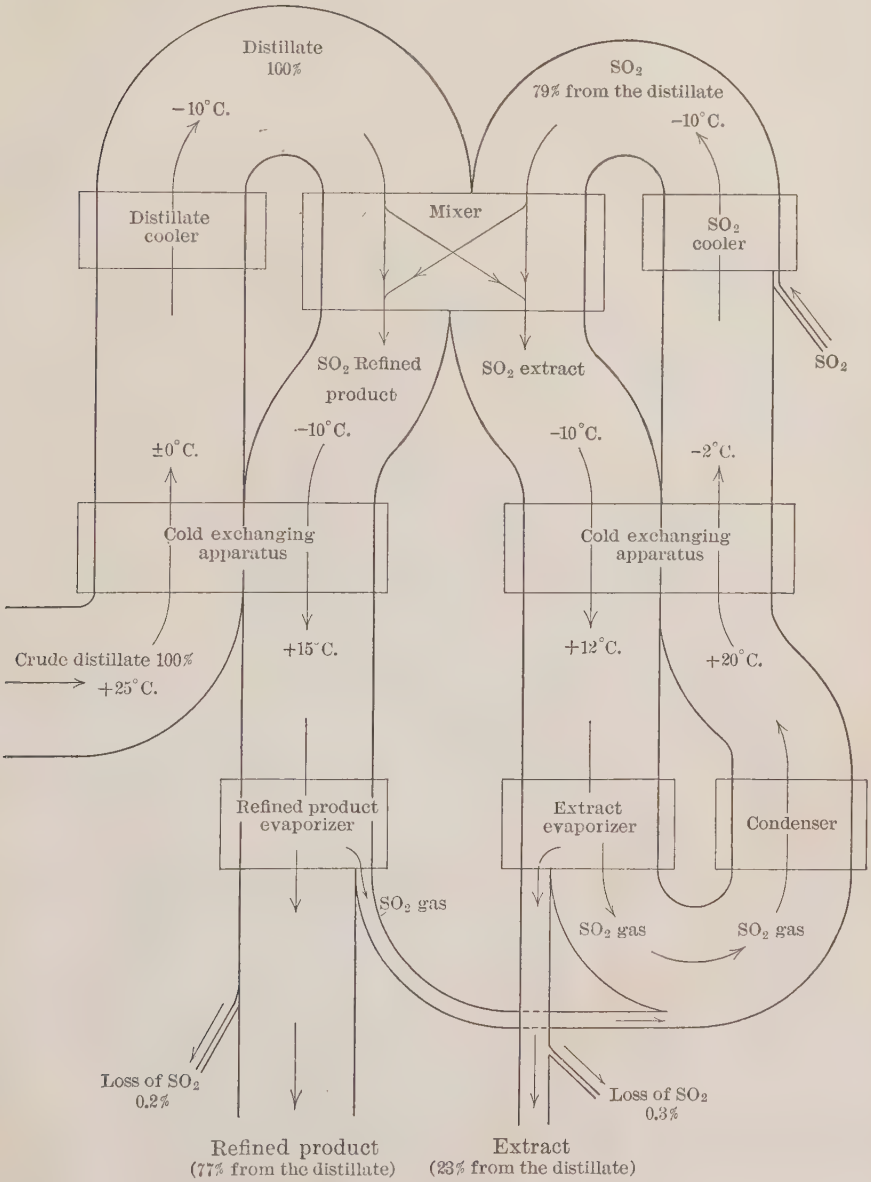


FIG. 6.—PLAN OF THE EDELEANU PROCESS.

The arrangement of the plant is such that from the operator's platform the coolers which are placed high and the evaporators which are

below, as well as the mixer in the center, may be watched and attended to. As can be seen in Fig. 7, all the important pipes with their control valves are placed on a regulating table, so that the operator in charge of the refining process can take care of all the necessary regulating and connecting manipulations without leaving his place of observation.

Working Results

The working results of a plant with an average daily capacity of treating about 65 tons of distillate per 24 hr., as were reached in the plant at Ploesti, are given in the following table:

Specific gravity of the crude distillate to be treated	0.820
Average time of operation, minutes	68
Quantity of distillate treated at each operation, kilograms	3,115
Temperature in the mixer, degrees centigrade	−10
Specific gravity of treated product	0.8028
Specific gravity of extract	0.8691
Sulphurous acid left in the treated product, per cent	0.16
Sulphurous acid left in the extract, per cent	0.36
Consumption of fresh steam per operation, kilograms	1,200
Consumption of fresh steam per 1,000 kg. distillate, kilograms	385
Consumption of exhaust steam per operation, kilograms	648
Consumption of exhaust steam per 1,000 kg. distillate, cubic meters	208
Cooling water used per operation, cubic meters	17.1
Cooling water used per 1,000 kg. distillate, cubic meters	5.5
Loss of sulphurous acid per operation, kilograms	17.5
Loss of sulphurous acid per 1,000 kg. (distillate), kilograms	5.6

From the above figures, the cost of treating a Roumanian Bustenari distillate can be calculated as follows:

Cost of Treatment in a Plant of 65 *Tons Daily Capacity*

Cost of Plant:		Francs
Machines and apparatus		130,000
Buildings, construction, freight, duty and other expenses		100,000
Total		230,000

Amortization and Interest:		Francs
Amortization	= 10 per cent. of 130,000	13,000
Amortization	= 5 per cent. of 100,000	5,000
Interest on the invested capital	= 5 per cent. of 230,000	11,500
Total		29,500

In 325 working days, at 65 tons per day, there will be 21,125 tons treated, so that the amortization and interest amount to Frs. 1,396 per ton, or per 100 kg. ..... Frs. 0.140

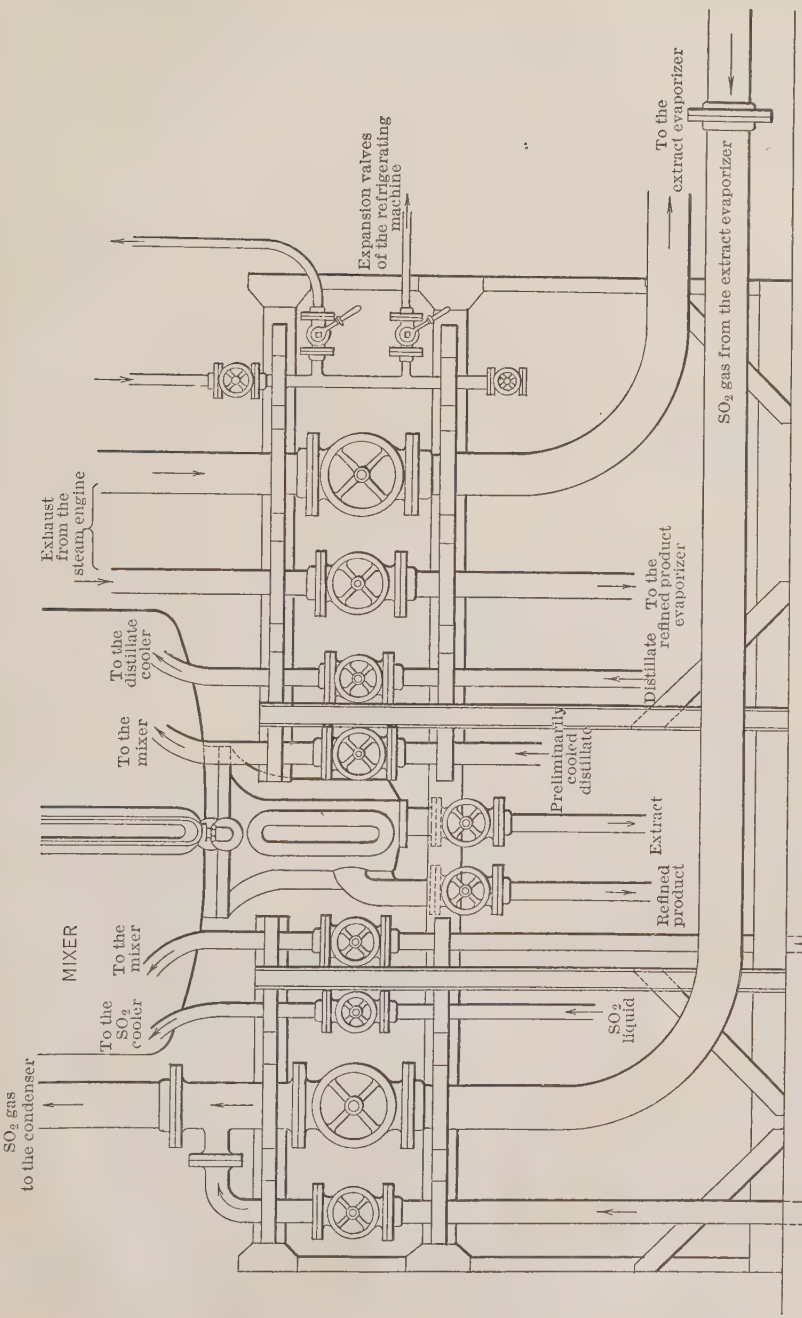


FIG. 7.—DIAGRAM OF REGULATING TABLE IN APPARATUS ROOM.

Maintenance, Labor, etc.:	Francs
Repairs, working materials	6,000
Wages and salaries	8,000
Insurance, 1½ per cent	3,450
Lighting and heating	2,400
Drying salt, unforeseen expenses	3,000
Total	22,850

The cost of maintenance and labor, therefore, amount to Frs. 1.082 per ton, or per 100 kg. ..... Frs. 0.108

Cost of Materials:

There are required for each 1,000 kg. of distillate:	Francs
385 kg. fresh steam at Frs. 3.50 per ton	1.35
208 kg. exhaust steam at Frs. 1.50 per ton	0.31
5.5 cu. m. cooling water at Frs. 0.04 per cubic meter	0.22
5.6 kg. sulphurous acid at Frs. 18.00 per 100 kg	1.01
Cost of materials per 1,000 kg	2.89
Cost of materials per 100 kg	Frs. 0.289

The total costs of treatment, therefore, amount to ..... Frs. 0.537 or practically Frs. 0.54 per 100 kg. of distillate or 3.9 shillings per ton. It must also be taken into account that Frs. 18 per 100 kg. of sulphurous acid is a very high figure and refers to imported sulphurous acid. The cost of this material is very much lower when sulphurous acid is used which is produced in the country itself, or actually in the refinery.

Cost of Treatment in a Plant of 500 Tons Daily Capacity

Cost of Plant:	Francs
Machines and apparatus	1,100,000
Buildings, construction, etc	500,000
Total	1,600,000

Amortization and Interest:		Francs
Amortization	= 10 per cent. of Frs. 1,100,000	110,000
Amortization	= 5 per cent. of Frs. 500,000	25,000
Interest on the invested capital	= 5 per cent. of Frs. 1,600,000	80,000
Total		215,000

In 325 working days, at 500 tons per day, there will be produced 162,500 tons, so that the amortization and interest amount to Frs. 1.323 per ton, or per 100 kg. ..... Frs. 0.132

Maintenance, Labor, etc.:	Francs	
Repairs, working materials	40,000	
Wages and salaries	30,000	
Insurance, 1½ per cent.	24,000	
Lighting and heating	10,000	
Drying salt, unforeseen expenses	20,000	
Total	124,000	
Brought forward		0.132
The cost of maintenance and labor, therefore, amount to Frs. 0.763 per ton, or per 100 kg		0.076

Cost of Materials:	Francs	
There are required for each 1,000 kg. of distillate:		
385 kg. fresh steam at Frs. 3.50 per ton	1.35	
208 kg. exhaust steam at Frs. 1.50 per ton	0.31	
5.5 cu. m. cooling water at Frs. 0.04 per cubic meter	0.22	
5.6 kg. sulphurous acid at Frs. 10.00 per 100 kg	0.56	
Cost of materials per 1,000 kg	2.44	
Cost of materials per 100 kg		0.244
The total cost of treatment		Frs. 0.452

or roughly, Frs. 0.45 per 100 kg. of distillate or 3.26 shillings per ton. In this case, the cost of sulphurous acid is taken at Frs. 10 per 100 kg., sulphurous acid produced on the spot being reckoned. The cost of production is amply estimated throughout.

The above tables are based on figures actually obtained from the existing plants. It is, however, to be anticipated that with further perfection of the working of the process the costs will be reduced even more.

In conclusion it may be well to summarize some of the chief advantages of this new process.

(1) The process makes it possible to obtain illuminating oil of good quality from the greatest variety of low-grade oils.

(2) The sulphur dioxide employed is cheap and practically all of it is recovered.

(3) Sulphur dioxide required for the process can even be obtained from the acid sludge available in the refinery in the manufacture of lubricating oils.

Discussion

F. W. Bushong, Pittsburgh, Pa.—This process of extraction is particularly advantageous in the case of Roumanian oil, on account of its high aromatic content. I have prepared a paper, which will appear probably next month in the *Journal of Industrial and Engineering Chemistry*, describing a series of two-degree fractions of oil from the Nowata field in Oklahoma. I have made extractions with liquid sulphur

dioxide in accordance with Dr. Edeleanu's method, and found that the Oklahoma oils, for instance the kerosene fractions, are purified in this way, although the quantity taken out does not at all compare with that taken out of the Roumanian oil. I have made careful analyses of separate fractions, before and after extraction, and they show a very distinct fractional separation. I like the distinction brought out by Dr. Day in previous discussions, in regard to oils, rather than oil, and his beautiful illustration of fractionation. Just as fuller's earth, and other clays, will enable us to obtain lower-boiling fractional portions of the oils, and eventually retain a residuum which is hard to get rid of, or to remove, so there is an advantage to be gained by the use of a reagent which begins at the other end by taking out these last constituents first, namely, ozone. I have also analyzed the ozonides and they are derivatives of hydrocarbons very much poorer in hydrogen than we have heretofore supposed. Similar results have been obtained with Roumanian, Russian, and Italian petroleums. Furthermore, I have prepared ozonides from a fraction of Oklahoma oils, both before and after extraction with liquid sulphur dioxide, and found that the ozonide obtained from the extract is the same as that precipitated out of the original oils. In other words, the residual substance, which the fuller's earth retains, is the one first thrown down by the use of ozone, but it may also be extracted by liquid sulphur dioxide and be thus separated in apparently unaltered condition. The analysis of this extract corresponds to the formula $C_{15}H_{20}$, which is surprising, since it is lower in hydrogen content than we have been in the habit of considering among the constituents of our oils. The extract was probably contaminated with other hydrocarbons, but I think it is unquestionable that we must look for hydrocarbons in petroleum which correspond to a formula at least as low in hydrogen as $C_{15}H_{20}$, and possibly even as low as $C_{15}H_{16}$.

The Capillary Concentration of Gas and Oil

BY C. W. WASHBURNE, WASHINGTON, D. C.

(Pittsburgh Meeting, October, 1914)

FORMER studies of sedimentary strata have been based upon the mineralogical and mechanical characters of the solid components, rather than upon the open spaces between them. For present purposes let us take another point of view, and consider not the rock material, but rather its cavities, the pores and the fissures or cleaved joints which connect with them. Let us think of the geologic column not as a series of rocks, but as a mass of irregular connecting capillaries. Different parts of the column may be distinguished by the absolute size and by the relative amount of its open spaces.

Porosity

Each stratum of the column is characterized by a certain porosity, which would be independent of the size of the rock grains if all were of the same shape. Slichter has shown that the percentage of open space in a pile of equal spheres is practically constant regardless of the diameter of the spheres. This is approximately true of clean, assorted sand or gravel, but it is far from being the case with most natural sediments, which have a widely variable porosity, the magnitude of which is determined mainly by the amount of fine and of secondary material between the larger grains. The shape of the grains materially affects porosity, round grains leaving more open space than angular or flat grains. The thin plates of kaolin and the fine plastic matter of clay fit quite perfectly when pressed together, and it is probably this close fit, determined by shape of grain, rather than the small size of grain, that causes the low porosity of ordinary clay shale.

The best-assorted, least-cemented sediments are the most porous. The porosity of even a very fine-grained rock may exceed 10 per cent. if the grains are fairly uniform and discrete as in the so-called "oil shale" discovered by Woodruff in the Green River formation of western Colorado and Utah. The rock has the megascopic appearance of a tough cemented shale, but it is really a consolidated calcareous mud or fine marl, composed of discrete, angular microscopic grains of calcite, with solid and liquid hydrocarbons in the interstices. The pores of this rock probably

are not large enough to be charged with oil by the method of capillary concentration described below, which moreover operates only in wet rocks, but they are much more efficient channels for migration than the pores of clay shale. If the rocks were dry, as they now are, at least near the surface, the fine pores of the limey layers would rob oil from fissures by capillary action, and thereby create the so-called oil shale of western Colorado. It is possible, also, that the hydrocarbons represent organic matter that was buried with the calcareous mud, and are therefore analogous in origin to the Scotch oil shale. The latter, however, contains no liquid oil until it has been heated in the retorts, and it occurs in a region where there are no oil seeps. The rarity of oil shales and their general absence from oil fields suggest an origin of their hydrocarbon content different from that of oil, or else that there is some process operating in oil regions which has destroyed pre-existing oil shales and which has concentrated the oil in the sands and fissures. The same reasoning shows that the porosity of a stratum can have no relation to the concentration of oil except in a passive way, being only a measure of the maximum reservoir capacity of the stratum.

Capillarity

Each stratum of the geologic column is characterized not only by a certain porosity but also by a certain average pore diameter, which may differ greatly from the average pore diameters of adjacent strata. The size of the capillaries varies approximately as the grain of the rock, being greatest in conglomerates and very coarse sands, and least in clay shales. Liquids are drawn into these pores by the force of capillary action, which varies directly as the surface tension of the liquid and inversely as the diameter of the pore. Crude oils probably have only about one-third the surface tension of water, and hence only about one-third of the capillary power.

Lord Raleigh[1] found that a water-air surface has a tension of 75.6 dynes per centimeter at 0° C. and of 72.8 dynes at 20° C. Somewhat lower values are given in recent tables, 73.21 dynes at 0° and 70.60 dynes at 20°.[2] A salty water, such as that commonly found in oil fields, having a density of 1.14 at 20° C., would have a surface tension of about 79 dynes per centimeter.[3]

[1] *Philosophical Magazine*, 1890; similar results by Hall, *Philosophical Magazine*, 1893.

[2] Landolph-Börnstein-Meyerhoffer tables, quoted by Müller-Pouillet, *Lehrbuch der Physik*, 10th ed., vol. iii, pp. 277 to 308 (1907). These are the same as the values found by Morgan and Stevenson, *Journal of the American Chemical Society*, vol. xxx, pp. 360 to 375 (March, 1908).

[3] Approximate calculation of the density of solutions may be made by the use of the *Smithsonian Physical Tables*, 4th ed., p. 91, and with this the surface tension may

No determinations of the surface tensions of crude oil have been found in the literature examined. The writer obtained a value of 24.1 dynes per centimeter at 20° C. by the capillary method, using Pennsylvania crude oil (specific gravity, 0.852). Müller and Pouillet[4] quote a value of 26.4 dynes at 25° C. for a (presumably refined) petroleum having a specific gravity of 0.847. Magie found a tension of 25.9 dynes at 20° C. on petroleum (refined?). Quincke[5] using a refined oil of 0.7977 specific gravity obtained a tension of 29.7 dynes on the oil-air surface and 28.9 dynes on the water-oil surface, both determined at 20° C.

It is highly probable that the molecular attractions produce a concentration of the fractions of lowest surface tension in the molecular "film" at an oil-air surface, and hence that the surface tension of crude oils is much less than would be expected from their specific gravities. For the same reason only pure material can be used in determining the surface tension of individual hydrocarbons even approximately. Some of these have been calculated by the writer from the values of the capillary constant α determined by Bartoli and Stracciati[6] and are given below. Evidently the old statement repeated in many text books,[7] that the addition of CH_2 does not change the surface tension, is not true for hydrocarbons, although it appears to hold for alcohols.

Capillary Constants of the Paraffin Series

Substance (Normal)	Temperature, Centigrade	α (*a*)	Surface Tension (*b*)	Substance (Normal)	Temperature, Centigrade	α (*a*)	Surface Tension (*b*)
C_5H_{12}	11.0	1.635	16.0	$C_{11}H_{24}$	14.0	2.688	26.4
C_6H_{14}	11.0	2.038	20.0	$C_{12}H_{26}$	12.8	2.781	27.2
C_7H_{16}	12.0	2.399	23.5	$C_{13}H_{28}$	14.0	2.848	27.9
C_7H_{16}	12.0	2.387	23.4	$C_{14}H_{30}$	13.0	2.915	28.7
C_8H_{18}	11.0	2.477	24.3	$C_{15}H_{32}$	13.3	2.997	29.4
C_9H_{20}	14.0	2.538	24.9	$C_{16}H_{34}$	14.0	3.041	29.8
$C_{10}H_{22}$	13.0	2.629	25.8				

(*a*) Milligrams per millimeter.
(*b*) Dynes per centimeter, calculated from (*a*).

At the end of this list there is an unfilled gap up to melted paraffin

be estimated from Volkmann's table, *Wiedemann Annalen*, vol. xvii, p. 353: *Smithsonian Physical Tables*, No. 142, p. 128.

[4] *Loc. cit.*

[5] *Poggendorff Annalen*, vol. cxxxix, and *Philosophical Magazine*, 1871, as corrected by Worthington, *Philosophical Magazine*, vol. xx (1885). See Gray, *Smithsonian Physical Tables*, 4th ed., p. 129 (1908). Worthington regards Quincke's values as too high.

[6] Engler, *Das Erdöl*, vol. i, table 22, p. 91.

[7] Wilhelmy, *Poggendorff Annalen*, vol. cxxi, p. 44 (1864). Jones, *Elements of Physical Chemistry*, p. 139 (1902).

which has a surface tension of 30.56 dynes at 54° C.[8] The surface tension of solid hydrocarbons, paraffin, asphalt, etc., must be very much higher, but small bodies of these, colloidal or other, would not affect the surface tension of crude oil. The unsaturated hydrocarbons appear to have a higher surface tension than the corresponding members of the methane series,[9] but if there is any of the latter in a crude oil it probably will determine the surface tension of the mixture. In general it follows from the Gibbs-Thompson law, that the hydrocarbons of lowest surface tension will tend to be concentrated at an oil-air surface, sufficiently to lower greatly its tension. Hence it is probable that all crude oils have low surface tension, probably in the neighborhood of 25 dynes, except only the oils that have lost all of their light constituents, which are of no consequence in problems of deep migration. It is unfortunate that these deductions must be based on theory. The need of accurate experimental work is very great.

Capillary Concentration

Since water has about three times the surface tension of crude oil, capillary action must exert about three times as much pull upon it. The amount of the capillary pull varies inversely as the diameter of a pore. Hence the constant tendency of capillarity is to draw water, rather than oil, into the finest openings, displacing any gas or oil in the latter. Gas itself is not drawn into capillaries by the action of surface tension, and it can leave the fine pores without any capillary resistance. It is therefore the most quickly and the most completely gathered in the largest spaces available. Moreover, capillarity resists any movement of water from fine toward large pores more than it resists the movement of oil or gas in that direction. In short, water enters fine capillaries about three times as readily as oil, and it encounters about three times as much capillary resistance in leaving them.

The final result of this action must be the concentration of nearly all the gas and oil in the openings having least capillary power, namely, in fissures if present, and in the larger rock pores. This appears to be a general rule in oil fields, where the pores of the coarser-grained rocks contain practically all of the oil, while the pores of the adjacent fine-grained rocks contain practically no oil. Any slow flow between shale and sand would bring about the concentration of oil and gas in the sand, and in the absence of such independent movements it is possible that

[8] Landolph-Börnstein-Meyerhoffer tables, Müller-Pouillet, *Lehrbuch der Physik*, vol. iii, p. 277 (10th ed.).

[9] Hexane has a surface tension of 20 dynes at 11° C. (Schiff); benzene, 28.3 at 11° C. (average of various authorities, except Morgan and Stephenson, who found a tension of 29.38 dynes at 22.5° C. or 30.5 at 11° C.); benzole has a surface tension of 29.86 dynes at 12.5° C. (L.-B.-M. tables) or 30 dynes at 11° C.

capillarity alone would drag enough water into the shales to displace most of the gas and oil. However there is no need to place such a heavy burden upon capillarity, since independent flows would almost certainly be created by the differences of pressure or head arising during the deformation and subsequent erosion of the strata, besides the general upward slow flow of the rock fluids that is indicated by the fact that the observed downward increase in pressure is greater than the corresponding hydrostatic head. Any of these movements of the rock fluid would concentrate gas and oil in the sands, regardless as to whether the movements are continuous across the sands or oscillating, because capillarity pulls water out of sand into shale with greater force than it exerts on gas and oil, and because capillarity offers greater resistance to the passage of water from shale into sand than it offers to gas and oil. The recognition of this principle is the first step in the clarification of the local lithologic distribution of gas and oil.

In dry rocks there could be no capillary concentration of oil. In the absence of water, oil would be drawn into the finest pores and would be widely diffused. It is probable, however, that the pores of all clay shales contain abundant free water, in addition to the adsorbed moisture, except in dry regions near the ground surface. A rock like the so-called oil-shale of western Colorado and Utah might conceivably be formed by the capillary impregnation of layers of fine porous limestone by oil which ascended through fissures into the dry rocks near the surface. Since the pores of the limy layers are as fine as those in shale the bituminous layers could not be formed by capillary concentration in wet rocks. On the other hand productive deposits of oil and of gas, which are limited strictly to coarse-grained rocks and fissures, cannot be formed in the complete absence of water, nor if formed could they long exist in dry rocks, because they would be destroyed by capillary diffusion into the finer pores.

There is both an upper and a lower limit to the diameters of pores in which capillary action can take place. The capillary action of pure water is nil in tubes over 0.508 mm. diameter, and in smooth fissures over 0.254 mm. wide. Openings larger than this exert no attraction on water, and when present they are the most favored loci of the capillary concentration of oil and gas, because water is the most effectively removed from them. Capillary action of an average crude oil is nil in tubes over 0.2 mm. diameter and in fissures over 0.1 mm. wide, yet openings of this class are favored in the concentration of oil, because water is the controlling medium, and it is drawn into the finer openings with about three times the capillary force of oil, which tends to be left behind in the larger openings. The water tends to displace the gas and oil in the smaller openings, shoving them into the bigger spaces.

The minimum diameter of capillary openings is somewhat uncertain.

It is usually placed at 0.0002 mm. for tubes and at 0.0001 mm. for plane fissures. Owing to the great friction in such small openings, the rate of flow is almost negligible,[10] and there is little need to consider their capillary action. According to theory, capillary action must cease when a pore becomes so small that there is not room for two or more molecules of water to pass through it together, that is, in the case of a fissure when its width becomes less than twice the radius of the sphere of molecular action. Quincke[11] finds that the range of molecular attraction of water is about 0.00005 mm. This indicates that the minimum diameter of capillary openings is 0.0001 mm. for fissures, and probably twice as much or 0.0002 mm. for tubes.

Capillary Pressure

The maximum theoretical pressures developed by capillary action are those developed in the finest tubes in which capillary action is possible. Johnston and Adams[12] have calculated these capillary pressures for the temperatures prevailing at various depths, but they probably err in extending their calculations to tubes as fine as 0.00001 mm. (0.01 micron), which gives 20 times the theoretical maximum pressures. The work of Quincke and of Lord Raleigh shows that there can be no capillary action in such small tubes. Accepting the assumptions of Johnston and Adams, and the minimum diameter of 0.0002 mm. for tubes, the maximum capillary pressures developed by pure water will have the values shown in the following table. The capillary pressures of a salt water such as that common in oil fields, would exceed these values by about 5 per cent.

Approximate Maximum Capillary Pressures

Diameter of Pores, 0.0002 mm. (0.2 μ).

Temperature Gradient, 1° C. per 30 m.

Depth, Meters	Pressure, Atmospheres	Depth, Meters	Pressure, Atmospheres
100	15.0	2,000	12.1
200	14.6	5,000	7.8
500	14.1	10,000	1.0
1,000	13.6	20,000	0.0

Assuming a temperature gradient of 1° C. per 30 m., as in the preceding table, it is evident that capillary action loses about half of its force at a depth of 5,000 m. Since the heat increases more rapidly downward in many oil fields, it is probable that the capillary force decreases half at depths of 3,000 or 4,000 meters, below which it becomes prac-

[10] See calculations by Johnston and Adams, *Journal of Geology*, vol. xxii, pp. 1 to 15 (1914).

[11] *Poggendorff Annalen*, vol. cxxxvii, p. 402.

[12] *Loc. cit.*

tically ineffective. Moreover the surface tensions of all but the lightest hydrocarbons decrease much less rapidly than that of water for each increment of temperature, so that the surface tension of water does not have such great excess over that of oil at these depths. Hence it is probable that the capillary concentration of oil and gas must all be effected within 4,000 or 5,000 meters of the ground surface. Oil in deeper strata must remain diffused in the shales, if that were its original distribution, unless it was concentrated in the sands at some former period when the strata concerned were closer to the surface.

It will be observed from the preceding table that even the maximum capillary pressures are wholly inadequate to account for the high pressures in oil and gas fields. Nor is it probable that capillarity contributes anything to the total pressure on a body of oil or gas, since the capillary pressure operates in the opposite direction, *i.e.*, toward the finer pores and is really a pull from the larger pores, or a negative pressure. It would decrease the pressure of the collected oil in the sand were it not for the return of fresh oil and gas displaced from the shale, which nearly counterbalances the partial vacuum that capillarity tends to make. Therefore it is necessary to abandon previous assumptions that capillary action may contribute to the high pressures in oil fields, the mysterious source of which must be sought elsewhere.

Possibly the high pressure is an effect of the upward migration of the rock fluids (mostly water, nitrogen and methane) which are subjected to outward pressures of deep-seated origin. If we accept the plausible assumption that the rock pores and fine fissures are continuous, and that they extend downward to the zone of rock flowage, it is apparent that this deep connection would cause the abyssal gases and liquids to press upon those above, raising their pressure above the normal hydrostatic pressure, and tending to crowd them toward the surface. An imperceptible outward translation of the rock fluids *en masse* would be sufficient to explain the excess pressures in oil fields. The capillary penetration of the abyssal fluids need not approach the base of the sedimentary strata to account for the increased pressure in sands far above them. However, there are some phenomena such as the excess temperature in oil sands,[13] the excess of chlorine in oil-field waters,[14] and the great amount of helium and argon in natural gas,[15] which suggest that the upward migration in oil regions may be more extensive than the excess pressure would seem to require, and that possibly some fluids of abyssal origin may be penetrating far through the sedimentary strata.[16]

[13] Höfer, Temperature in Oil Regions, *Economic Geology*, vol. vii, No. 6, pp. 536 to 541 (September, 1912).

[14] Washburne, Chlorides in Oil-Field Waters, *Trans.*, xlviii, 687 (1914).

[15] Cady and McFarland, *University Geological Survey of Kansas*, vol. ix (1908).

[16] Moreau and Lepage argue the abyssal origin of the nitrogen and associated rare

Time

In tubes approaching the minimum capillary diameter the rate of flow from capillary action would be negligible. But in pores of 0.001 mm. diameter, which is of the same order of magnitude as the intergranular pores in shale, the capillary movement of water would be about 15×10^{-6} c.c. per year.[17] In a rock containing 10 per cent. of pore space, this would result in a flow of about 15 c.c. across each square centimeter per year. Probably half of this amount, or about 7 c.c. per square centimeter per year, corresponds more closely to the factors of flow imposed by the porosity of ordinary clay shale. This amount of flow seems quite adequate to produce capillary concentration in a geological period, but not in short time. In 10,000 years the movement, if continuous, would amount to 700 m. However the movement would not be continuous, but would vary in direction, with the changes in pore diameter, etc., and only small distances could be traversed directly by capillary pull. The actual amount of capillary movement required rarely would exceed a few meters, since most shales are cut by fine fissures, in which the oil would first be collected by capillary exchange for water. In big shale formations the fine capillary fissures probably gather the oil from its original disseminated distribution in the shale, and pass it on to the coarser joints and pores of the sandstones. The fine fissures of shale offer much less resistance to the flow of oil than the rock itself, through which the flow need be only that required to reach a fissure. Under the principles here advanced, the capillary flow of oil in wet rocks should tend always toward the larger openings.

Moreover, it is not necessary to assume that all of the movement in the pores is due to capillarity. Doubtless there are other slow motions of the underground fluids arising from various causes. Any motions of the rock fluids, whether continuous or intermittent, direct or reversed, or oscillating, would assist in the concentration of oil and gas in the larger openings, because water always would enter the finer capillaries with greater ease than oil or gas and would leave them with greater difficulty. The more to-and-fro movement of these fluids between shale and sandstone, or between shale and fissures, the more perfect will be the capillary concentration of oil and gas in the sands and fissures.

A factor which tends to increase the perfection of this concentration is found in the behavior of water vapor, which is evaporated from surfaces of large curvature at the same time that it is condensed upon surfaces of sharp curvature. In this way water is passed from the coarse pores of sandstone, and from fissures, to the fine pores of shale. The

gases in coal mines and in thermal springs. *Academy of Science*, Paris (Mar. 23, 1914); abstract in *Revue Scientifique*, 52e année, p. 442 (1914).

[17] Johnston and Adams, *loc. cit.*, p. 14.

vapor transfer will begin as soon as capillary concentration has partly filled the sand with gas, and thereafter it will supplement that process. This may explain the dryness of natural gas.

Extended Application. Dry Sands

Capillary concentration must operate on any of the gases present in the rock pores. These consist of nitrogen (which commonly is distinguished from atmospheric nitrogen by its relatively high content of argon and helium), of methane, and of carbon dioxide. Oxygen probably does not occur in the deep gases except in rare traces, and samples containing it probably are contaminated with air. The capillary concentration of nitrogen in sands is exemplified by the great nitrogen well of North Dakota, and by the gas of the western fields of Kansas, which in places contains over 80 per cent. of nitrogen that is notably rich in argon and helium. Gases rich in carbon dioxide are rare, outside of California, and the disappearance of the great quantities of this gas which would be formed by the decomposition of organic matter, is an unexplained difficulty of the organic theories. All of these principal rock gases are concentrated in the coarser pores by capillary action.

Capillary concentration does not of itself produce pressure. Many of the deep sands of the Appalachian and other regions are apparently dry, yet they do not contain enough gas under pressure to cause much escape when they are penetrated by the drill. Doubtless the sands were once full of the water in which they were deposited, but now they absorb some of the water used in drilling. Their dryness may be explained by the process of capillary concentration, as well as in other ways.

The process should operate effectively in all sands in which there is no very active supply of water, but not in sands like the Dakota of most Western localities, where there is an artesian circulation that would overcome the capillary withdrawal of water. In sands of this type the process would be effective only in places which are protected from rapid water circulation by structural peculiarities or other causes.

Weakening of Water "Films"

There are two main elements of possible weakness in the preceding hypothesis: First, the weakening of the capillary force of water by contamination with oil, and second, the high resistance of bubbles to capillary flow. A thin film of oil or other greasy substance, amounting to only an invisible trace, greatly lowers the tension of a water-gas surface, reducing it almost to that of an oil-gas surface. However, there is considerable tension on the water-oil surface,[18] which in the case of light

[18] Quincke found a tension of 28.9 dynes per centimeter on a surface separating water and petroleum. *Poggendorff Annalen*, vol. cxxxix, as corrected by Worthington, *Philosophical Magazine*, vol. xx (1885).

oils, is nearly equal to the tension on an oil-gas surface. In a capillary tube the water-oil surface is concave toward the oil. In other words, a water-oil surface exhibits the same capillary phenomena as a water-air surface, the difference being only one of degree. The capillary force of a water-oil surface is decreased very little by the slight angle of contact of the surface with the pore walls, and it amounts approximately to one-third that of a water-air surface. Hence the capillary behavior of water and oil is nearly the same as that of water and air, and there is a constant tendency for water to be pulled into fine capillaries previously occupied by oil. This tendency is demonstrated experimentally by the familiar damage to a lamp flame produced by a little water at the bottom of the lamp's reservoir. If the wick touches the water, the latter rises through the capillaries previously filled with oil, makes the flame splutter, and often extinguishes the light. In the same way water will pass from the coarse spaces of sand or from fissures into the fine capillaries of shale, displacing the oil, which is thereby forced into the sand through neighboring larger pores. Thus, the weakening of the water-gas surfaces by contamination with oil, is counterbalanced in part by the capillary action of the water-oil surfaces.

Effect of Capillary Gas Bubbles

The second element of possible weakness in the hypothesis lies in the high resistance to liquid flow offered by bubbles of gas and by drops of an immiscible liquid. If a fine capillary tube be filled with water containing bubbles of air, so that the bubbles are strung along the tube in alternation with drops of water, a pressure difference of over one atmosphere is required to produce flow in the tube. The various water-air films appear to cling rather tenaciously to the walls, and each seems to take up a part of the external pressure by a slight change of curvature. The action of the pressure is analogous to a vertical pull or push on a horizontal weighted cable supported at the ends. A certain amount of energy must be expended in lifting or depressing the central part of the cable, and in changing its form of curvature, before its terminal supports are affected. A series of drops of oil or of gas in water, is divided by twice as many catenoidal "films," stretched across the capillary, among which the applied pressure is distributed, so that the total resistance of a long series of such drops in a fine tube is very high.

Rock pores, however, are not single tubes, but the most irregular passages between the grains of rock, having countless lateral and longitudinal connections by means of which many of the little water bodies communicate with each other. They become single bodies as regards their tensional forces, which tend so to draw them together as to reduce the total area of their surfaces. The minute bodies of oil and of gas have

a similar tendency when connected in any way, however devious. All the drops and bubbles of the three fluids must grow into larger, more compact, bodies. Thus a condition of many small alternating bubbles of gas and drops of liquid could not exist very long in the rock pores, and they could offer only temporary resistance to capillary flow.

Geologists sometimes appeal to hypothetical emulsions as a means for the migration of oil. This is wrong. For the reasons stated above minute bodies of oil can have only temporary existence when compared with the long time required for geological processes. Surface tension will soon destroy an emulsion of water and petroleum, and moreover the high viscosity of emulsions would prevent their migration through fine pores. There is no reason for believing in the existence of any natural oil emulsions before the final rush of the oil and water out of the rock pores into an open well.

The Anticlinal Theory

The rôle of capillarity in producing the anticlinal distribution of oil requires consideration.

Whenever light oil touches a water-air surface, it is immediately spread over the surface and retained by surface tension. A heavy oil is retained with nearly equal strength, but after spreading a molecular film of its lightest elements over the surface of the water, it gathers itself into a drop, or remains as an irregular thick mass. If a bubble of gas rising through water touches a globule of oil the two immediately unite, and the oil forms a thin continuous shell about the gas bubble. Johnson has suggested that this process may explain the accumulation of oil at the top of water-soaked anticlines, the oil having been lifted bit by bit on bubbles of gas.

The theory is plausible and better than the old assumption that the difference in density between water and oil caused particles of oil to rise to the highest saturated parts of sands. It should be noted, however, that the volume of gas required is many times (probably 100 times) the volume of the oil which it could lift in this way; also that small masses, such as bubbles or drops, do not have sufficient defect in weight to be forced upward through fine pores. Before gravity can move them it is necessary for the globules of oil and gas to be united into larger masses by surface tension as described above. Not until then can the lifting force on a body of gas or oil be sufficient to cause its ascent.

It is probable that very large bodies of oil must be gathered before they displace sufficient water to receive a pressure competent to force them upward against the friction of irregular rock pores. A very simple demonstration of this fact may be made. Take a basin of water, and with a pipette or fountain-pen filler squeeze drops of oil against the bottom and sides of the vessel. It will be observed that most of the drops

cling persistently to the vessel and do not rise through the water. The force holding them is not surface tension because the same surface energy prevails when the drops are free. Gravity does not dislodge the drops, which can be dislodged only by stirring the water or by tapping the vessel. The force holding the drops is merely the friction of contact, yet if one of the same drops was spread out in the pores of an ordinary oil sand, the area of solid contact and therefore the external friction would be from 50 to 200 times greater. The gravitative lifting force on each drop would be the same as in the open vessel, and conclusively it could not move the drop of oil against the greater friction in the pores of a sandstone. This confirms the contention of Munn that gravity is not competent to cause the accumulation of oil above water in anticlines, etc.

The following working hypothesis is suggested to explain the common anticlinal distribution of oil.[19] First, the oil and gas must be gathered in the sands by capillary concentration. Second, the oil and gas must be segregated by the effects of surface tension into bodies large enough to be lifted by gravity, *i.e.*, into bodies so large that the difference between their weight and that of the displaced water is sufficient to drive them upward through the pores of the sand. Whether the volumes of such bodies would be a few liters or many cubic meters is a point requiring further study, but they are certainly of much higher order of magnitude than the drops of oil of the preceding experiment.

It is improbable that surface tension can gather oil into bodies large enough for the preceding requirement. Therefore a third process or stage is now postulated. Gas rises through water-saturated sand probably 300 to 500 times as readily as oil.[20] This may account for the fact that the tendency of gas fields toward anticlinal distribution is much stronger than that of oil fields. It is therefore much easier to assume that gravity can collect gas at the top of anticlines than it is to assume the less probable condition that it can lift oil through the rock pores to a similar position.

After a small reservoir of gas has been started at the crest of an anticline, then whenever circulatory or migratory movements of the water body bring a particle or mass of gas or oil in touch with the water-gas surface, the gas or oil can never descend. They will be held on the water-gas surface by its tension. The oil will first make a film along the top of the water, and any oil which is brought in contact with this film must

[19] Other working hypotheses are under consideration.

[20] This approximation is based on the fact that the lifting force of water displacement applied on a given volume of gas would be about 7 times the lifting force on the same volume of oil, combined with the moderate assumption that oil would suffer from 40 to 70 times as much frictional resistance as gas. Data for exact calculation are not at hand.

join it. In this way a deposit of oil will be formed between the water and the gas, and neither the ascent of the oil nor its larger lateral collecting movement in the sand need be the result of its low specific gravity. Any movement of the water in the sand, which would in time bring the various parts of the water in contact with its upper surface, would carry bodies of oil of all sizes to the water-gas surface, where surface tension would retain them permanently.

As an example, imagine a slow lateral migration of the water in a gently folded sand. Such slow migration has probably occurred in many oil sands, because of the commonly evident differences of head. Gravity would cause the accumulation of gas in the sealed arches, and these accumulations would not be destroyed if the sand be thick and if the flow of water be very slow. On the contrary, the transported accessions of gas would increase the deposit. Because of surface tension, the transported masses of oil would cling to the water-gas surface whenever they touched it, and later contributions would cling to the water-oil surface. Thus an anticlinal deposit of oil would be formed without the intervention of gravity except in the initial collection of an anticlinal body of gas.

These principles hold generally under any theory of the origin of oil. The process of capillary concentration is especially required by the organic theory, under which it is necessary to postulate an originally disseminated distribution of the oil in shale or limestone. Supporters of the organic theory may claim that the present absence of oil in the pores of shale adjacent to the oil sands is due to its removal by capillary action. Under the inorganic hypothesis it is not necessary, as a rule, to assume that the oil ever has entered the fine pores of shale. Its entire migration may have been in fissures and sands, and it may never have been widely disseminated. All of these possibilities must be considered until someone shall find satisfactory evidence concerning the organic or inorganic origin of oil.

Conclusion

The final condition of capillary stability under which the rock fluids come to rest is not an equilibrium of pressure, but rather a condition of minimum surface energy. Balanced pressures would be secured by the occupation of the finest openings by gas and oil and of the coarsest by water. Capillarity, however, does not operate in this way. It seeks only to reduce the surface energy to a minimum by the shortest route, which consists in the destruction of emulsions, the union of bubbles and of drops of like substances, and in the movement of the substance of highest surface tension (water) into the finest capillaries.

Capillary concentration gathers gas and oil in the coarsest spaces available: In fissures and super-capillary spaces when present; in conglomerates or coarse layers in sandstone; in sandstone layers in shale; in

dolomitic and cavernous parts of limestone. It cannot operate in dry rocks.

Surface tension collects the small bodies of gas and oil in a sand into larger bodies, which in the case of gas are capable of gravitative displacement by water, and are lifted to the highest position they can reach. When circulatory or other movements of the water bring occluded masses of oil in contact with the water-gas surface, the oil is held firmly on the surface by its tension. In this way a deposit of oil is gathered at the top of the water-bearing part of the sand. In the absence of these conditions, oil could accumulate below water, as it does in some fields.

The capillary concentration of oil and gas in sands and fissures is an essential requirement of the organic theories, which postulate that oil was originally disseminated in shales and limestones. It is not essential to the inorganic theories, under which the phenomena of concentration are capable of independent explanation, but the principles stated are thought to hold true for all oils in porous rock, regardless of their origin.

Discussion

Roswell H. Johnson, Pittsburgh, Pa.—It seems to me rather important to bear in mind one thing which is sometimes lost sight of, that the marine sedimentary rocks are laid down in water. We have to start with an initial charge of water. That being so, and the surface tension between the water and the oil being as slight as it is, as referred to by Mr. Washburne, the phenomena of capillarity are bound to be most important as between the liquid surface and the gas surface. So far as the liquid-gas surface is concerned, capillarity, in my opinion, is of the utmost importance, and it seems to me this paper of Mr. Washburne's is of great value to us for that reason. Starting out as we have, with our attention concentrated upon gravitational sorting, we have had to wait for the discussion of movement furnished by Munn, and this discussion of capillarity by Washburne, for a more balanced view of the situation. It seems to me that in the accumulation of oil and gas we have a problem of the interworking of gravitational sorting, capillarity, viscosity, immiscibility, and movement. All five things seem to me to be important. I would have expected Mr. Washburne to have given a fuller treatment of the relation of movement to capillarity. In my own experiments I find that gravitation and capillarity alone seem remarkably impotent when everything is at rest, but if I furnish a little movement, gravitation and capillarity step in and accomplish a great many results that one would not expect. Take a horizontal tube and fill a portion at the middle with an ordinary sand saturated with oil, and then put water and sand in each end, and the tube may remain at rest a very long time without that oil distributing itself along the top of the tube. But with a

little motion that oil will gradually work up to the upper side. So gravitation and capillarity alone are rather impotent unless they get this element of movement. It would not be necessary to have very extensive movement, but some degree of movement is essential.

Now if natural gas, in addition to the method of origin by the action of bacteria on organic matter in mud, may also arise by pressure at great depth, as the overburden is added, a great deal of motion may result. I think more motion is produced in this way than by the descent of meteoric waters.

The author, I think very properly, calls attention to the fact that acceptance of the rôle of capillarity still leaves us using the anticline and the structure contour in prospecting. It seems to me very important that gas men should realize that their relation to the anticlinal guidance is on a different basis from that of the oil man. If anticlinal prospecting has a certain degree of validity for oil, it has a far higher degree of validity for gas, so that the absolute necessity for gas men taking into consideration the anticlinal structure seems to me very important.

When I wrote the article referred to by Mr. Washburne, I was skeptical for the reasons given as to whether the difference between oil-gas and water-gas surfaces in surface tension could be of much importance. Yet, having observed in the laboratory how a drop of gas in water becomes so quickly surrounded by oil if there is any oil at hand, and strongly believing in the great importance of capillarity as between fluid and gas, a method by which capillarity would indirectly affect oil was suggested.

There is another practical consideration in prospecting that depends upon this theoretical question. Does oil arise in sand or in shale? These views indicate very strongly in my mind that there is no longer that necessity for its arising in the sand that we have had pressed upon us by some authors, particularly English. If the oil has to be indigenous in sand, the best section for our prospecting would be one with a maximum of it. We should have a large percentage of sand contributing to our ideal section. On the contrary, our best oil fields always have a great preponderance of shale, with merely interpolated beds of sand.

D. T. Day, Washington, D. C.—I think it is perfectly clear that the study of Mr. Washburne, concerning the accumulation of oil by capillarity in the earth, views the matter somewhat as a wick connecting that oil with something else: Oil will flow through and accumulate at the other end of the wick. But such an accumulation would not develop any particular pressure. Mr. Washburne has hardly laid stress enough on the very great pressure which may result where the oil has been driven out of a shale by the superior capillary force of water. In that way a very great pressure can be obtained.

Mr. Washburne's paper, following a great deal of practical work that has been done by a number of persons, including myself, is a very long theoretical step in advance, which will serve the particular purpose of clearing the atmosphere and developing a theory which will be easy for others to think of and to apply practically. It will stimulate a great deal of practical experimentation, I hope.

I object to the way in which he has used the expression "friction" in his paper in several places. This idea of friction seems to be more exactly adhesion, than friction, which we must think of as always a function of motion. It is a function varying with the speed with which particles are moved; when the speed is zero, friction is zero. The average geologist forgets that he speaks of friction where the motion is infinitely slow and friction cannot exist.

This paper also should receive a modification, I think, in your minds in this way. The geologist is not necessarily a chemist, and the geologist will consider oil as oil. There is no such thing as oil. You must think of *oils*. And the very first thing that happens in the migration of oil through the earth is practically always a change in the character of the oil itself. That offers complications to this theoretical consideration. It also simplifies the matter, inasmuch as by the careful chemical study of these oils one can derive a great deal of information as to where the oil has been, and its relation to the other oils nearby. The study of earth movements of oils is going to be very much simplified by determining the differences in oils adjacent to each other. For example, in about 1896 it was my privilege to determine for the first time, I believe, that mineral oils can be changed in composition by diffusion through rocks, especially shales.

By taking a Pennsylvania oil, and carefully filtering it until it was a perfectly clear liquid, but still crude oil, I absorbed it in dry clay, practically such a plastic clay as fuller's earth. Shale from the region of Titusville, Pa., was also effective. The clay was then shaken up with water. Water drove out most of the oil but not all. It did not drive out the oil that went in, but it drove out different oils. Is that perfectly clear? You drive out some oil if you put in an amount of water equal to the volume of clay. If you put in more water, you will drive out more oil, but when you are through, oil will be left in the clay that cannot be driven out by the water. I do not say that in geological time we might not drive out every particle of that oil, but I do not believe so, and we have no proof that we can. If, instead of clay, you take finely powdered glass and absorb the oil with that, and treat that with water, you get nearly all the oil out in the same condition in which it went in. Two features are essential to the discussion of this matter: Mr. Washburne's article considers oil and space, no matter what the wall of the space is; the chemical nature of that wall is not considered. Neither is one oil

considered as different from any other oil. Oil is oil. Those two things are fatal for a complete discussion of the theory of this subject, because one oil will work in one way, and another oil will work entirely in another way. In the first place, the kind of surface of the capillary tube exerts an important influence other than simply on the rate of capillary movement. That is shown best in this way: Suppose you allow an oil to be soaked up by a clay. Now add water and examine the oil that comes out, and examine the oil that is left in the clay. There are means by which you can get it out, but not by simple addition of water to it. Take those two oils. What are they? If you determine what proportion of the oil that went in was made up of unsaturated hydrocarbons, and determine the proportion of unsaturated hydrocarbons in the oil driven out by the water, you will find that the oil driven out by means of water has a much less percentage of unsaturated hydrocarbons in it. If you go further and take the clay, from which all the oil that is possible has been driven by water, and try to dissolve out the oil with gasoline, some more of the oil can be dissolved out, but not all of it. By adding ether, some of that oil which is perfectly soluble in ether can be taken out. But ether will not take out all the oil which was perfectly soluble in ether before it was absorbed in the clay. Go one step further and try benzol. That will drive out some more, but not all of it. And as far as I have gone, I do not know of anything in the way of a solvent that will extract all of that oil out of the clay, although the oil was perfectly soluble before.

That just indicates that very possibly some of those shales which are supposed not to have any oil in them, because it cannot be extracted by the ordinary solvents, but only by destructive distillation, may still have oil there as such, in the same condition as the oil which has gone into a clay and cannot be brought out by means of any ordinary solvent. In other words, from the evidence of some 2,000 experiments carried on in this study by myself, it is perfectly evident that so soon as an oil containing mixtures of saturated and unsaturated hydrocarbons goes into a clay (not powdered glass, but clay) the capillary surfaces of clay act differently in respect to different oils. As soon as an oil goes into clay *fractionation takes place.* The unsaturated hydrocarbons cling to the surface of these capillaries and will not go on. The saturated hydrocarbons, interdissolved at first with the unsaturated hydrocarbons, go farther and pull away from the unsaturated material in solution with them. That holding back of a material by a surface has received a particular term that is not at all explanatory. It is simply naming that property as adsorption—not absorption. That word "adsorption" is used by Mr. Washburne here apparently meaning absorption, the two being confused by himself, or possibly by the printer. By adsorption these things can hang back. We have a familiar example of that in the manufacture of lake dyes. If an analine dye is dissolved in water and

alum added, a clear solution of the analine dye and alum in water results. That dye will remain dissolved indefinitely. But as soon as ammonia is added, aluminum hydroxide is formed, which takes hold of the dye and leaves an absolutely clear water solution again, the dye entirely disappearing. This is an analogue of the property of clay surfaces of taking one oil out of solution in another, and holding it back.

This must be considered in connection with problems in the earth. We must remember that there are certain things in the way of oils that cannot go through shale at all. An asphalt oil from Texas will not pass through a shale until that shale has been entirely saturated with asphalt. Then the other light, unsaturated hydrocarbons will simply hang back until the clay surfaces are saturated by them and then finally they go through and are not affected by these surfaces, except in respect of capillarity, as with saturated hydrocarbons.

We hope to simplify our problems very greatly by the study of the capillary concentration of oils. We must give careful consideration to the study of all these capillary phenomena, especially to the adsorptive effects on certain hydrocarbons, which undoubtedly have played a great rôle in this matter.

Turning to Mr. Johnson's discussion: he has enunciated a very important idea in regard to movement. It seems to me that when we look at the relations of oil and gas and the great deal of help effected by movement, we recognize the very considerable difference in solubility of natural gas in various oils. The old idea of a layer of water and a layer of oil and a layer of gas in contact with each other must be greatly modified, because under the pressures existing most of that natural gas must be in solution in the oil. A bottle of Vichy looks clear, but when the bottle is opened the gas escapes. It was there in solution in the water. So we have a larger reservoir of gas for a given pressure if the oil is present to dissolve that gas. In the case of oil with gas in solution, just as soon as the pressure on the oil is released the oil remains but the gas escapes. Any change of pressure on that oil must affect the gas in solution in the oil, so the cause for the movement of oil, in migration, is, it seems to me, perfectly self-contained. Any slight change of pressure or temperature will be sufficient to give plenty of reason for movement.

Edwin M. Chance, Wilkes-Barre, Pa.—I am very glad Dr. Day touched upon the subject of adsorption. That, to my mind, is the most important consideration of this subject. Mr. Washburne in his paper has considered the action of the capillaries as purely mechanical. I am afraid that if we consider it from this angle only, we will see but a very small part of the picture. The nature of the capillary surfaces, as Dr. Day has well said, is of great importance, as the phenomena of adsorption will exercise, and, I believe, have exercised a great and far-reaching

influence upon the nature, chemical composition, and general characteristics of the crude oils. The phenomena of adsorption are of every-day occurrence and very wide usefulness in the arts. The clarifying action of bone black, fuller's earth, and like substances, depends entirely upon this action. The clarification of drinking water by alum precipitation depends largely on this, and it is only reasonable to believe that when an oil penetrates gradually through great areas of rock, taking great time, and where the rocks are of a shaly nature, closely allied to what are known to be the best substances for adsorption phenomena, this transfusion must greatly affect the nature of the oil.

H. A. Wheeler, St. Louis, Mo.—The complications which Dr. Day's experiments certainly show are extremely important and are complications that vary with every clay; for I think Dr. Day will indorse my statement that you never find two clays alike in this respect.

To revert to this proposition solely as a mechanical factor, Mr. Washburne's paper would make the hydrostatic pressure entirely a factor of porosity. I would like to ask the members present if in a new field, before there has been any loss of pressure by drainage, the rock pressures have ever been found equal to or greater than the hydrostatic head due to the depth of the sand? In my own experience, I cannot recall a single instance where the pressure was not more or less under that which would be expected from the mere hydrostatic head; sometimes quite a marked shrinkage, which, with all due respect to Dr. Day, I will call friction loss.

Roswell H. Johnson:—There is one reason why the hydrostatic interpretation of gas pressure is not wholly acceptable. As we go down in the sands we seem to get excesses of gas. Particularly in this field it becomes excessive in quantity relative to the amount of water in the sand. If this holds in other fields as well it must mean that there has been an ascension of water, because these sedimentaries, when laid down, were filled with connate water. If we do get an ascension of connate water, produced by some such phenomena as its expansion by heat or by the formation of new gas by pressure upon organic materials at great depth, we can hardly expect the pressure to be controlled merely hydrostatically.

* In reply to Mr. Wheeler's inquiry as to gas wells with pressure higher than hydrostatic pressure, I would cite the original wells given by Orton on which he based the hydrostatic theory. He used in his calculation a head of 600 ft. above tide, assuming the outcrop of the Manitoulin Islands to be in Lake Superior, whereas the Trenton does not outcrop at Lake Superior. The Manitoulin Islands are in Lake Huron, the level of which is 580 ft. above tide.

* Communication to the Secretary.

No allowance is made for the lower density of the fresher water near the outcrop, the weight taken for the whole column being the water of high density found at the wells.

While Dr. Orton admits that "defects of porosity would abate the pressure" he assumes no such "defects" from the gas wells to the outcrop, a violent assumption, since Phinney states that the Trenton is "close or only feebly porous" over large areas and Bownocker refers to a well in Wood County that had no gas until a depth of 50 ft. into the Trenton, although most of the production was obtained above that horizon. He also states that "the pay commonly ranges in thickness from 5 to 30 ft., it is more porous and commonly has a darker color than the inclosing rocks and hence is easily recognized."

Another objection to Dr. Orton's calculation in this case is, that in order to get the proper pressures to appear for Indiana, Phinney takes his head from the Trenton at the Cincinnati anticline rather than Lake Superior. If for the Trenton in the Indiana field, why not for the Trenton in the Ohio field?

G. A. Burrell, Pittsburgh, Pa.—Some years ago I made a few experiments at the request of Dr. Day, in which I dissolved natural gas in crude oils from California, to saturation. I found that at ordinary pressures and temperatures the oils absorbed from 15 to 25 per cent. of their volume of the natural gas of Pittsburgh. Casing head gases were absorbed to a greater extent, as much as 60 per cent. Under heavy pressures, of course, the solubility effect would be much greater.

I have been interested in the subject of the movement of oil and gas through the strata, principally in the movement of natural gas. The two are much related. We found that natural gases vary much in composition, as many others have noticed. We have, for instance, the so-called dry gas, or marsh gas, usually not found in localities where oil is present, at least usually not in as tremendous volumes as the gases that are present in the oil fields. In the oil fields we usually have a gas containing a higher percentage of the heavier paraffin hydrocarbons than methane. We were able to separate those hydrocarbons in the case of Pittsburgh natural gas and found that it contains 84.7 per cent. of methane, 9.4 per cent. of ethane, 3.0 per cent. of prophane, and 1.3 per cent. of the higher paraffin hydrocarbons, chiefly butane. That is typical of the great quantity of gas used in Pittsburgh, Columbus, Cleveland, Cincinnati, and other places that obtain gas principally from the fields of West Virginia, though a great deal comes from Pennsylvania, and some from Ohio.

Passing from that type of gas to a gas in contact with oil in the stratum, we have found that the higher members of the paraffin series, higher than methane, predominate at times, and some of them contain

enough of the vapors of the liquid paraffins to make them profitable as a source of gasoline.

Referring now to the movement of gas through sands. We have taken samples at different places in certain localities from different sands—that is, from a lower sand and an upper sand—and from the few data we have collected, we have found that the gas in the upper sand contains less of the heavier paraffin hydrocarbons than that in the lower sands, looking as if in moving from a point of common origin and passing up through strata and finally landing in the top strata, the gas had lost some of the more easily removable paraffin hydrocarbons.

We ran across an interesting case in the natural gas from the Hogshooter pool, Oklahoma. That is a natural gas, containing about 95½ per cent. of methane, about 3½ per cent. of carbon dioxide, and the rest nitrogen. An unusual condition surrounding the gas lies in the fact that there is a tremendous body of gas containing methane as the paraffin hydrocarbon not far from oil sands where other paraffins than methanes are found.

If these gases came from a place of common origin with the other natural hydrocarbons, here is a case where in moving through a comparatively short distance, a tremendous quantity of gas had lost its higher paraffin hydrocarbons.

I wish to lay emphasis on the relation between the composition of natural gas and the presence of oil. As we find natural gas in the oil fields, it almost invariably contains the higher paraffin hydrocarbons, and the inference is that natural gas issuing from earth containing methane as the only paraffin hydrocarbon, may be usually taken as an indication that oil is not present in the immediate vicinity. In rare cases this may not follow.

D. T. Day, Washington, D. C.—Our experimental data may be of great value perhaps in one connection. It has not been enough to go far on the chemical side of this question. It is principally on the geological side. We recognize that where oils contain unsaturated hydrocarbons they are, as a class, taken out by adsorption.

But suppose you only have paraffin hydrocarbons, what happens when they go through this clay? In the ordinary course of diffusion, the more viscous saturated hydrocarbons hang behind, so that you do get fractionation of paraffin oils. It is perfectly easy to show this by experiment. Taking a tube 6 ft. long, pack it tight with fuller's earth, put one end, open, into a receptacle containing oils from which all unsaturated compounds have been very carefully removed, and allow that oil to diffuse slowly through the clay and you will get gasoline at the top and vaseline at the bottom. You get it by a process that is purely diffusion. In this case there is no action of adsorption, so far as we know. But you do get fractionation by diffusion.

Professor Gurowicz, in commenting on the work I did in 1897, states that while I did effect those results I did not know what I was doing, and did not know why I got the results. He seems to think that there is no other action than adsorption. In the above case there was no separation of any class of hydrocarbons; there was purely diffusion, having reference to the viscosity of the different oils. He does not consider the diffusion idea at all. He thinks it is entirely adsorption. I am prepared to combat that view. What Mr. Burrell has said here is sufficient evidence that that is going too far. Here, with a mixture of very volatile hydrocarbons, such as butane, propane, ethane, and methane, we evidently have separation by diffusion of those perfectly saturated hydrocarbons, so that eventually you have nothing but methane. It seems to be a perfectly clear case of separation by difference in viscosity. Just exactly as the original separation of hydrogen from oxygen by heat was made by the superior diffusibility of hydrogen. If you can fractionate two gases in that way, so you can fractionate two oils. This fractionation which Mr. Burrell speaks of seems to me to be exactly of the same class.

F. W. Bushong, Pittsburgh, Pa.—When one takes an ordinary steel cylinder and fills it with a natural gas in the field, at high pressure, one finds on first opening it a very large excess of methane. The mixture follows the natural law of gas passing out through a small orifice. In that cylinder there will remain finally the high-boiling constituents. I wonder if Dr. Burrell has not observed that same thing in the passage of gas through the porous strata of the different fields, and I wonder whether this is not also a factor in the matter of hydrostatic conditions. The gas, of course, under high pressure will be dissolved in much larger quantities. But the point I want to ask him is whether that same field examined later on, after the enormous pressure has been relieved through consumption, will not show eventually a large amount of casing head gasoline and, like the Ohio and Indiana fields, will not yield almost entirely gasoline, where it now yields almost entirely pure methane.

G. A. Burrell.—In the Hogshooter field the pressure is down pretty low; starting at 500 or 600 lb., it is now down to about 30 lb. I would think that if there was going to be a change in composition, it would have been noticed by now.

F. W. Bushong.—Have you observed such a change?

G. A. Burrell.—Yes, that change takes place. The lighter paraffin gases have passed off in large measure in many oil wells.

C. W. Washburne, New York, N. Y. (communication to the Secretary*).—This reply to the discussion of my paper at the Pittsburgh meeting is based on the stenographer's transcript received in Africa.

* Received Mar. 6, 1915.

I thank Dr. Waldo for his offer of new data on the surface tension of crude oils, which I hope he will add to this discussion at an early date. His observation that crude oil has surprisingly low surface tension, below what would be expected from its specific gravity, is in accord with my determination of the surface tension of a Pennsylvania crude oil, mentioned in the paper. It accords also with my deductions from capillary theory.

I agree with Mr. Johnson in everything he says. The necessity for other movement, independent of capillary action, in order to enable the latter to effect concentration, is recognized in the paper, but the greater emphasis he gives the matter is helpful to correct understanding.

Mr. Johnson is correct also in objecting to the opinion of Höfer and other European geologists who believe that oil must originate in the same sands in which it is found and that it cannot penetrate shale. Clay is much better than sand for the preservation of organic matter and the great bodies of shale and limestone furnish a more adequate organic source of the oil which saturates the sands. Certainly in fissure fields, such as the Florence field, Colorado, and probably in some of the smaller Wyoming fields, in which there are no sands, it is absolutely necessary to admit that *the oil has passed through shale* in order to reach the short discontinuous fissures, or gashes, in the great shale formations. If the fissures at Florence reached the underlying Dakota sandstone, the field would have been flooded with water when the pressure was exhausted. It remains remarkably dry.

With Dr. Day I must differ. As shown in the paper, it is highly improbable that capillarity can cause any effective pressure whatever in the sands. The pressure in the closed tube, from which Dr. Day had driven the oil with water, may have been due to capillarity, but the mechanical conditions of the closed tube are not imitated in nature, except in fine passages that open into coarse ones, and which are closed at the other end. Capillarity would thus create pressure in the closed pores of shale opening into sand, but not in the sand.

Probably it is necessary to increase my estimate (15 atmospheres) of the maximum pressure which can be developed by capillary action. In the paper I called attention to the fact that "the minimum diameter of capillary openings is somewhat uncertain. It is usually placed at 0.0001 mm. for tubes and at 0.0002 mm. for plane fissures." These are the values given in Van Hise's *Treatise on Metamorphism*, in text-books, etc., based on the work of Lord Rayleigh and others, and are the basis of my calculation of maximum pressure. However, I have received a letter from John Johnston, of the Geophysical Laboratory, Washington, D. C., calling attention to the more recent work of G. Bakker[21]; who considers

[21] *Zeitschrift für physikalische Chemie*, vol. lxxx, No. 2, p. 129 (June 25, 1912).

that the radii of the spheres of molecular attraction are only six to seven times the molecular diameter. Consequently the length of the radii is only a few millimicrons, and capillary action can take place in pores very much smaller than 0.0002 mm., developing pressures far in excess of those stated. This entirely justifies Johnston and Adams[22] in calculating capillary pressures for tubes as small as 0.01 micron, and invalidates my objection thereto. I have not had opportunity to examine the articles by Bakker (the only literature here being a missionary's bible!), but in this study it is not necessary to pay further attention to the thickness of capillary films and to the corresponding possible pressures in minimum capillary tubes, because the capillary pressure would be effective only in closed passages in shale and could not contribute to the pressure in the sand. Only a negligible pressure could be developed in the comparatively coarse pores of sandstone by the capillary withdrawal of water from fissures, and this would be more than overcome by the negative pressure left in the sand pores by the more general capillary pull of the water into shale. The capillary pressure of the basal water against the oil in the sand pores also would be negligible. There is no volumetric change involved.

Water would be drawn into the minimum capillaries of shale with great pressure, amounting to many hundred, possibly to a thousand, atmospheres, judging from Bakker's calculation of the sphere of molecular influence mentioned above. In rock pores the actual pressure would be of course very much less than that in the theoretical minimum capillaries. The pressure would be effective in driving the oil out of the minute pores of shale into joints and fissures and into the comparatively coarse pores of sandstone, but the attempt to explain pressure in the sand by such means is like the problem of a man lifting himself by his boot straps. Likewise, the attempt to explain the general migration of oil by capillarity or any continuous capillary movement, or any ascent of a liquid above the capillary height for the given pore diameter, is the problem of perpetual motion. These fallacies must be abandoned by students of oil.

My use of the term "friction of contact" was rather careless. Dr. Day is right in saying that friction is not merely adhesion, and that it increases with velocity, but he fails to note that in the little experiment in question it was necessary to consider only one element of friction: namely, the adhesion of drops of oil to the sides of an open vessel. This may be regarded as the initial element of friction, since it is the resistance to the first movement of a drop. This element is independent of velocity, and depends upon the area of solid contact, which in the pores of ordinary sands would be from 50 to 200 times the area of contact of the drops against the sides of the vessel. All other elements of friction, both internal (viscosity) and external, which come into play as soon as a drop begins

[22] *Journal of Geology*, vol. xxii, No. 1, p. 13 (Jan.–Feb., 1914).

to move, are to be added to the initial adhesion, and they strengthen the argument that gravity cannot lift drops of oil through a fine water-saturated sand. Gravity possibly may lift larger masses of light oil, as in Johnson's experiment, mentioned in this discussion, after circulatory or other movements of the water body have dislodged them, but not little drops, nor heavy viscous oil. Where the latter has been gathered in low anticlines, as in the Kern River field, the oil probably has increased afterward in gravity and in viscosity. The circulatory movements of the water would dislodge all of the drops in time, and these circulatory movements are much more efficient than gravity in carrying the drops and other oil bodies to the top of the water, where they would be held by the tension of the water-gas surface, whenever they happened to touch it. In very fine sands the process would be less efficient because of the small radius of curvature of the water-gas surface in the little pores.

The time available for the concentration of oil into sands may be very long, but it is not infinite. The time available for producing the anticlinal and other types of distribution of oil commonly is very short. The relation of some oil bodies to the ground surface and to the ground water indicates that the present distribution of the oil has been effected since the erosion of the existing topography. "Motion infinitely slow," therefore, would be ineffective, as it is in most geological processes, and hence in this study we cannot agree with Dr. Day when he says that here "friction does not exist" and that "it has reached the minimum limit." We cannot neglect friction in the study of the migration of oil, and I greatly regret the lack of data concerning the factors of flow for oil in small passages. The viscous lag of the heavier paraffin hydrocarbons in Dr. Day's experiments must have been due to friction. If friction ceased, there could be no viscous lag in geological migration.

Dr. Day then proposes a modification of my paper by adding another subject, the fractionation of crude oils by migration through shale, a very useful discovery of his, which, however, does not strictly concern the subject of the paper, capillary concentration. This is purely a mechanical subject and I would prefer to leave chemical and other extraneous matter for other discussions. Capillary action is independent of the nature of the capillary walls and depends only on surface tension and on pore diameter.

Fractionation by filtration is due to adsorption and to viscous lag, and these factors, especially the former, are strongly influenced by the nature of the capillary walls, but this side discussion can receive little attention here. The word "diffusion" has no rightful place in the discussion of the filtration of liquid hydrocarbons. Diffusion is a molecular process which operates only in gases and in substances dissolved in liquids. To say that since oxygen and hydrogen may be separated by diffusion, therefore diffusion may separate a mixture of two liquid hydrocarbons, violates the

principles of molecular physics, unless the reference be only to the vapors of the hydrocarbons, which evidently is not the case. All the phenomena of fractionation by filtration may be explained without calling upon any unknown or mysterious molecular process, simply by noting the controlling factors, which are: first, the selective adsorption of the unsaturated hydrocarbons, of the sulphur compounds, and possibly of others, upon the surfaces of clay particles, and secondly, the progressive frictional lag of the more viscous fractions.

I regard the first factor, adsorption, as the most important feature of filtration in nature. The effects of the second factor, viscous lag, are thought to be blotted out in the later stages of migration, when the delayed viscous elements follow the lighter ones and reach the same destination. This is indicated by the experiments in Engler's laboratory, in which the filtration was not stopped as soon as the tube of fuller's earth was saturated with oil, but the oil was allowed to continue flowing through the earth, whereupon it was observed that the succeeding later fractions became heavier.

However, when we take up the subject of fractionation, it will be necessary to consider the other alterations of crude oil, especially oxidation and polymerization, and the catalytic influence of clay; also the reasons why Arnold finds that the oil of California gets heavier as it migrates. At the first opportunity I will present my field observations in support of the essentials of Dr. Day's theory, also the chemical and geological matter concerning the modification of oil during migration and during its long storage in the sands. In this manuscript, which I left practically complete in America, I have shown that crude oil grows heavier with time, and that probably all crude oils are descended from lighter parent oils. I regard partial oxidation, polymerization, and other molecular combinations, as the controlling factors in the alteration of oil in nature.

The adsorption of the unsaturated hydrocarbons on clay, as in the experiments of Day, Gilpin, Cram, Bransky, and others, removes the more condensible elements of crude oil and renders it less susceptible to polymerization and to chemical change. For this reason, combined with the protection given by the thick bodies of shale, Pennsylvania oil is lighter than that of the limestone fields of Ohio. In both States, as well as in California, and elsewhere, the distribution of the oils of different character indicates that they are the condensed or polymerized products of oils which were originally much lighter. The viscous lag of the heavy fractions, as in Day's experiments, cannot have much geological importance, being only a temporary phenomenon.

In reply to Mr. Wheeler, he will find the first discussion of an observed pressure greatly in excess of the possible hydrostatic pressure, by consulting Orton's early description of a big gas well in New York State. Since

then similar observations have been common. They appear to be especially numerous on the Apscheron Peninsula and in lower Louisiana, coastal Texas and Mexico, fields in which the communication across the deeper strata appears to be especially open, judging from the geological structure, from the locally high temperatures and generally high pressures, from the abundance of the chemically active gas, H_2S, with some CO_2 which has low diffusive power, and from the unfiltered character of the oil, which is asphaltic. Most of the oils are highly asphaltic and viscous, except in some of the Baku and southern Mexico fields, where the oil contains little asphalt, but is nevertheless rich in unsaturated hydrocarbons. This is the general character of the fields along the coast of the Gulf of Mexico, which are broken by faults, dikes, volcanic necks and salt plugs. On account of these vertical passages, there is little difference between the coastal fields that develop the middle Cretaceous limestone, and those that develop sands, dolomite lenses, and fissures in the overlying shale or marl. As soon as one passes inland to the interior field of Mexico, discovered by the writer, and to the northern fields of Louisiana, which are *in strata of the same age, but unbroken*, the fields acquire an entirely different character. All of the preceding characteristics are absent, there are no great gushers nor notably high temperatures, the gas is largely methane, and the oil is of filtered type, it is light paraffin oil. Asphaltic oil and hydrogen sulphide is found only below the Eocene marl or shale, in the underlying middle Cretaceous limestone. In these interior fields the absence of strong cross fractures forced the oil and gas to filter through the shale, probably making its way largely through minute joints. Since adsorption depends mainly on the area of clay surface over which the oil moves, there is no reason why oil should not be filtered in passing through the fine joints of a thick mass of shale, although we can admit a small amount of seep through the body of the shale. In this filtration into the interior fields of Mexico and Louisiana, the unsaturated hydrocarbons were removed by the clay, leaving the oil incapable of much polymerization, and therefore it remains comparatively light. The less diffusive and chemically active gases could not pass through the shale.

With further reference to Mr. Wheeler's question, Munn and others have shown that large areas of the deep sands of West Virginia and Pennsylvania are dry. As stated by Johnson in this discussion, the dryness suggests that the pressure has forced the water out, the pressure being therefore something that opposes the hydrostatic pressure. This explanation now appears as good as any, but in the paper there are other possible explanations of the dryness of the deep sands. The very fact that the sands are dry shows that they do not contain the water required to account for the pressure, which in the gas sands approximates that of the hydrostatic pressure corresponding to the depth of the well, apparently regardless of the elevation of the outcrop. There are also some notable depar-

tures both above and below the theoretical hydrostatic pressure. Since the deep sands are largely dry, the common approximation to hydrostatic pressure can have only one explanation, which is, that the pressure is applied through the water in the pores and joints of the overlying shales and other strata.

This in itself is proof that shale is not entirely impervious. The minute joints and pores of the shale must communicate, doubtless very imperfectly, but nevertheless with sufficient freedom to permit the transmission of pressure through their liquid contents. If pressure can be transmitted through the shale from above, it can be transmitted equally well through the rocks below. Hence there is no reason to doubt that the pressure of the abyssal fluids may be transmitted outward through the fine rock pores and joints. The work of Adams[23] has shown that these passages may remain open at far greater depths than was formerly thought.

I infer a small, but locally active, very slow, outward creep of the rock fluids (liquid or gas or both) due to pressure from below. The principal reason for this inference is based on plotting the distribution of pressures in the sands of different fields of the United States and of Canada for which data are available. The data are not sufficiently complete to warrant final conclusions, but there are so many instances in which the downward increase of pressure between sands, and the increase above the surface pressure, exceed the corresponding hydrostatic head, that I have been led to believe in the very slow upward movement of water, and probably of oil and gases, across the strata; also that gas pressure is in part of abyssal origin. It seems significant that excess pressures are most common in the more fractured regions, such as Baku and the Gulf Coast, where the deep communication appears to be more open than elsewhere. Other arguments for this ascent are found in the excess temperatures in oil fields, in the distribution of oils and gases of filtered and unfiltered types, in the excess of chlorine in the associated water, and in the abundance of helium in the deep wells of Kansas.

On account of its great abundance, an atmospheric origin of the helium in Kansas gas seems out of the question. Moreover, the percentage of helium in the nitrogen of Kansas gas increases with depth, pointing toward a deeper source.[24] The marked tendency of radium emanations to collect on clay suggests that possibly the helium may have been derived from emanations, which could more readily penetrate the rocks, but if this were its origin we would expect to find very high radio-activity, which appears not to be the case. The helium probably has risen by diffusion across the strata from a deep internal source, possibly magmatic.

[23] *Journal of Geology*, vol. xx, No. 2, pp. 97 to 118 (Feb.–Mar., 1912).

[24] Cady and McFarland: *University Geological Survey of Kansas*, vol. ix (1908).

The common approximation of gas pressure to hydrostatic pressure may be assigned to the fact that the weight of the water in the rock pores resists the slow outward movement, and that equilibrium is established when the outward pressure is equal to this weight; *i.e.*, to the local hydrostatic head. In such cases the hydrostatic head may be regarded either as the cause of the pressure, applied in most cases downward through the overlying shale, rather than through the sand from its outcrop, or it may be regarded as a measure of the principal resistance which is met by the deep-seated pressure applied from below. Where there is equilibrium this is immaterial, but where there is departure from the theoretical hydrostatic head the latter interpretation becomes necessary. The cases of excess pressure may be assigned in general to greater freedom of communication underneath the "sand," as in the domes of the Gulf Coast, to locally more active ascent of the rock fluids from any cause, or to greater resistance above the "sand." The cases of original defect in pressure below the theoretical hydrostatic head may be assigned, in general, to the opposite causes, or to leakage, or to dryness of parts of the overlying rock. It is necessary to remember that the phenomena of oil fields are exceedingly variable, that no two fields are alike. Doubtless there are many fields in which the hydrostatic pressure of the water in the sand, below the outcrop, controls the pressure, but in the Appalachian, Rocky Mountain, and Mexican fields this explanation is generally impossible.

In this matter a word of caution is necessary against the acceptance of pressure readings with the Pitot tube as giving anything more than partial approximation to the true original pressure in the sand. The Pitot tube gives the flowing pressure at the casing head, to which must be added the loss of head (pressure) required to force the gas rapidly through the pipe and the much greater loss by friction in the sand. Since great gas wells rarely can be closed completely until some time has elapsed, it is necessary to rely on the Pitot tube for pressure measurements, but we lack data for calculating the pressure in the sand ("closed" or "rock pressure"[25]) from these measurements. The writer has failed in the attempt to make such calculations, and he will be greatly indebted to any mathematically inclined student of gas who will develop a formula or table for the purpose, which is applicable to the variable coarseness and porosity of sands.

These questions demand separate papers. In the present study pressure need be considered only in its relation to capillarity. I believe I have shown that capillarity is wholly incapable of causing pressure in the sands.

The word adsorption occurs only once in the paper and is correctly

[25] The gas man's term "rock pressure" should be avoided in scientific discussions, since it may be taken to mean pressure exerted by the rock. "Closed pressure" is preferable.

used if printed with a "d",[26] since it refers to the moisture adsorbed on clay surfaces as distinguished from the free or capillary water. Dr. Day's objection to the use of the word is not understood.

Mr. Burrell infers an upward creep of natural gas from his observations of the downward increase in the percentage of heavy hydrocarbons in the gas of different sands. This new argument tends to support my hypothesis of the outward creep of fluids, and I beg Mr. Burrell to give us this useful information in greater detail. Has he observed any downward increase in the percentage of CO_2 and of other less diffusible gases?

[26] I have not seen the paper in print, nor the proof, having only a copy of the manuscript.

The Oil Fields of Mexico

BY EZEQUIEL ORDOÑEZ,* MEXICO CITY, MEXICO

(Pittsburgh Meeting, October, 1914)

I HAVE read in the *Bulletin* a paper by H. von Höfer relating to the Origin of Petroleum,[1] in which the author supports his and Engler's views, expressed before, of the organic origin of petroleum. Von Höfer pronounces strongly against the hypothesis of the volcanic origin of hydrocarbons, as particularly advocated by Eugene Coste.

In Mr. Coste's articles, so well known by the public, he refers incidentally to our Mexican oil occurrences, especially of the Gulf coast, as proofs of the volcanic origin of the oils. This belief comes from the fact that, in our Mexican oil fields of the east coast, the majority of the producing wells are located near volcanic necks (remnants of small volcanoes, mainly of the explosive type), or near certain gibbosities in shales, resembling mounds, some of which are really uplifts due to volcanic forces or to abortive eruptions.

The first attempt to explain the oil coming up around volcanic necks in Mexico, was made by me in a small article[2] published in 1905, soon after the investigation ordered by the Mexican government as to the outlook of the Mexican Gulf coast as a future oil producer.

When E. L. Doheny of Los Angeles, Cal., and I located the first producing well of Mexico, I was also the first to predict the great future of the oil industry in Mexico, just at the time when the government and some Mexican geologists felt rather pessimistic. The Mexican investors and the general public, as a result of these discouragements, did not pay attention to this important source of common wealth until recent days. The realization of the value of this source of wealth has been so sudden that we are menaced by the consequences brought about by a great boom.

It was quite impossible, about five years ago, to induce Mexican capitalists to invest money in oil lands. So the few large foreign companies operating since the beginning, easily secured large holdings, in spite of the trouble frequently resulting from faulty titles or ownership.

The main object of this short paper is to expound the fact that the connection between oil and small volcanic apparatus is exclusively

* Honorary Member.

[1] *Trans.*, xlviii, 481 (1914).

[2] Sobre algunose ejemplos probables de tubos de erupcion, *Memorias de la Sociedad Alzate*, vol. xxii (1905).

mechanical, having nothing to do, of course, with the origin of the oil, which is hypothetically considered by us as of purely organic derivation. Our oil is not indigenous to the rocks in which are contained the porous seams where it is accumulated.

As is well known, the Mexican oil lands are made up of a thick shale formation intercalated in places—especially near the oil seams—with a kind of sandy shale and thin hard flinty beds of the same material. These 4,000 or 5,000 ft. of shale overlie probably a thick Cretaceous formation of limestone with other sedimentary rocks intercalated, in which probably the oil originates.

While it is inferred in von Höfer's article referred to (in quoting the views of a Mexican geologist), that the oil country is a highly disturbed one, I think it is the contrary. That section of the east coast of Mexico does not show signs of great movements. The shale formation—as seen in cuts made by rivers, or in places where the gravel and residuary material of an eroded peneplain which mantled it are washed away—shows only slight gradient undulations, monoclinal low ridges and ample vaults, never big or repeated foldings, nor great displacements by faults. The latter, according to my general observation, are usually of very small importance.

Dikes are practically unknown in the central oil lands, between the Tamesi and Tuxpam rivers, and besides the necks and small lava streams there are some rare hills of plutonic rocks like diorites and nepheline syenites. Further north in the State of Tamaulipas and south to the middle part of the State of Veracruz, some sierras breaking up the coastal plains present more complex geological structures.

Local interior areas, very much disturbed, certainly exist, but have not been proved productive as yet. Highly disturbed shales are found also as a belt in the western margin of the formation against the underlying limestone of which the foot hills and the eastern Sierra Madre mountains are built up. The volcanic necks are seen scattered in the coastal plain commonly as isolated cones or as small groups of hills; the first type is wonderfully represented in the section between Chicontepec and Tuxpam, of the State of Veracruz.[3]

We should not make the volcanic action responsible for all the undulating and gently boss-like structure of the large shale oil-bearing formation, for everything tends to prove that the main tectonic, weak as it is, is prior to the volcanic activity in the coast. The volcanic plug or pipe of ashy and compact basaltic lava, of which the neck is the upper end, has not produced any considerable disturbance of the shale around it, and I figure that its passage through the sedimentary rock was like a screw through a piece of wood. I imagine that, during the making of the pipes and the

[3] Oil in the State of Veracruz, *Mining and Scientific Press*, vol. xcv, No. 8, p. 247 (Aug. 24, 1907).

coming of masses of hidden intrusive igneous material, there was a certain absorption of the sedimentaries by the igneous molten material in the depths, also the breaking of the sedimentaries near the intrusion and finally a narrow aureole around the plug of brecciated or broken material caused by friction, which zone is utilized in many cases for the rising of minute quantities of oil to the surface and the formation of seepages of very slow accumulation.

A number of large seepages, near which important discoveries have been made, are found at the foot of the necks between the shale and the basaltic rock; other seepages lie in the flanks of dome-shaped bosses. In no rare cases dry hard cakes of *chapopote*, with a soft freshly-risen heavy oil in the center, are encountered in the bottom of small amphitheaters, or horseshoe-like encircling spaces, or in general in the concavities made by curved rows of volcanic hills and small ranges. These different cases of seepages are very favorable for locating wells, as proved by experience (Cerro de la Pez, Chijol, Juan Casiano, Cerro Azul, etc.). Numbers of small seepages or *chapopoteras* are found in the middle of the coastal plain far from any salient topographical accident. It is frequently observed then that the oil exude is not coming directly from underneath the place, as seen from the surface, but that it has run, sometimes for considerable distance, on the shale between it and the thick argillaceous material covering that rock.

In the present state of our geological knowledge of the oil lands it is difficult to explain some important productive areas near river banks which have no apparent connection with volcanic outbreaks nor with bosses. Unfortunately, data collected during drilling and observations of the experts in this and other particular cases never come to the public domain.

I have seen many cases of very small exudes dropping slowly from narrow cracks and little faults in the cliffy banks of rivers where the shales are cut by erosion. This is the case in several places on the rivers Tecolutla, Tuxpam, Calabozo, Pánuco, Tamesí, Valles, etc.

The unimportant veins of albertite and grahamite, so well known in the coast, proceed probably from old exudes filling cracks and fissures where the *chapopote* dries slowly, becomes a little oxidized and is then subjected to pressure.

Lighter oil than that commonly found on the coast central oil lands appears to be found at shallower depths in the disturbed shales near the limestones at the foot hills of the eastern Sierra Madre mountains. This seems to coincide, as in Aquismon, with local impregnation of the limestones by light-oil products. In this case the oil probably comes up through the contact between the shale and the limestone which lie in unconformity.

While in this upper belt of the oil country conditions are still entirely

unknown, it seems to the writer that a possible source of petroleum exists in this long belt, under conditions probably different from the areas already known, or in the way of development, nearer the Mexican Gulf shore, or in the estuaries bordering it.

I have carefully studied, under the microscope, the sandy sediment accompanying the oils coming from several wells; especially from those near Tampico. This sediment is composed of minute shale splinters and *rounded* grains of basaltic material, similar to the rock of the necks. The volcanic origin of certain mounds and the influence of volcanic action, as facilitating the accumulation of oil, is demonstrated by what we have seen of volcanic rock particles brought with the oil from wells where no necks or any other volcanic rock exposures are found in the vicinity.

The 4,000 or 5,000 ft. thickness of uniform shale oil-bearing strata is naturally very impervious. We often have to drill hundreds of feet in entirely dry shale even in close proximity to the sea and many feet below the ocean level.

If, as we suppose, the shale formation is not very much disturbed and there are no real porous continuous strata in the formation, we have to believe, as far as our incomplete knowledge goes to-day, that the porous seams, afterward impregnated with oil at the common depths of 2,500 to 2,700 ft. at which we have generally found the oil, are made by volcanic action. Masses of strata corroded by molten lava extend to this depth. It is evident that the strata which become porous ought to show a particular physical structure and a somewhat particular chemical composition different from the rest of the bedding, which facilitates both corrosion and partial absorption by the hot molten lava. We imagine the oil seams elongated horizontally like sills of large dimensions. In order to show the capacities, let us consider a seam or a group of seams capable of producing 3,000,000 barrels without interruption, as in the case of the first producing well of Mexico at Cerro de la Pez near El Ebano, or of the famous burned well of Dos Bocas.

The peculiarities of the strata as supposed to exist, while of a nature absolutely unknown to us, must exist, since there is in our oil-bearing strata something like an oil horizon at an average depth of 2,600 ft.

The exhaustion of a good producing well is marked by the presence of abundant salt water, mud, and hydrosulphides. The porous spaces, after giving the oil, are probably refilled partly with salt water coming up to the seams by the old deeper conduits. The ascension of the oil through the drilling holes is caused by abundant high-pressure gas; the yield of the wells is generally intermittent. There are gushers of very long life. I cited several times the first producing well of La Pez which produced about 1,500 barrels a day for nearly 70 months.

Particular difficulties for rapid successful drilling are to be encountered in northern Tamaulipas and south Veracruz, as well as in the east coast of

Oaxaca and Tabasco, for the geological conditions of the shales are dissimulated by thick cappings of river and marine gravels in the north and muds in the south covering the vast peneplain extended in the isthmian country. The particular location of the Isthmus with regard to the rest of the eastern coast of Mexico and the Yucatan peninsula has created peculiar climatic conditions facilitating new detrital masses, favored by abundant rains, winds, heat, rivers, etc.

Discussion

David T. Day, Washington, D. C.—Dr. Ordoñez is really the pioneer in the study of the occurrence of Mexican oil. His studies came at such an early time as to prove really prophetic. He was successful in pointing out the relationship of these oil deposits to the volcanic teeth which are so very characteristic as a geological feature in the coastal plain of Mexico. At that time only the first wells around Ebano had been drilled, and drilled under the direction of Dr. Ordoñez himself. Only these Ebano wells were available for a real study of the condition of the occurrence of oil, the other large gushers not then being known. From that time on, the study of the geology of the country has, in large measure, been able to keep up with the oil development. As usual, a great deal of haphazard work in the exploration of the oil fields has been carried on. The results have been good, bad, and indifferent. We know a great deal more about the distribution of oil in Mexico than we do about the geology of it. Two ideas have become prevalent in regard to the Mexican oil, due to the peculiar development there: (1) That oil is omnipresent over a tremendous area bounded by De Soto La Marina, in the province of Tamaulipas, on the north, to south of Tuxpam in Veracruz; and (2) that there are very few spots in all this region which have rich gushers. The truth is somewhere between the two. The amount of land taken up, chiefly by three or four large companies, is extremely great, and covers all the possible chances. The idea that oil is going to be discovered in a large percentage of the land taken up by the oil companies, seems to me entirely unjustified. The amount of oil that will be produced from comparatively small spots here and there over that territory and other territory, undoubtedly will be very great. I do not think there is any hope of multiplying the capacity of these large gushers by the area of that part of Mexico in which they are contained, as a large number of promoters are inclined to do.

Another view is that these tremendous gushers will last a short while, then each well will be drowned out by salt water. Many people have really condemned a large and profitable field on account of that. Considering these areas from the north southward, the first large area is the Panuco region, where the Corona well of the Dutch syndicate is found.

Nobody has much idea of what that well yielded before it became convenient to measure it. The large pond of oil which this Corona well yielded was measured with a great deal of froth on it. The oil is very heavy, contains some water and froths a great deal. There is very much doubt that the product went up to 200,000 bbl. a day, as some claimed. Its yield was probably somewhere between 30,000 and 50,000 bbl. a day. This Panuco region has been very prolific in large gushers and gives every evidence that an area of at least 10 square miles will be quite productive. At least 2,000 acres in that region can be considered as proved oil territory of great value. No trouble has been experienced with water, but the oil is very thick. It is a peculiar class of oil, and will be valuable when a large fuel-oil trade is developed and facilities for handling it are afforded. But there is more oil now than there are facilities or market for it. East of Panuco is another interesting region, the Topila field, about half way between Tampico and the Panuco field. There, also, some good gushers have been discovered, but because of the influx of water have been prematurely condemned. That is unfortunate, because nearly every one of those gushers was very profitable before it went to the water. Yet an acre of that region is valued at perhaps one-tenth or one-twentieth part as much as in the Panuco region. Considering the price, I believe more oil will be produced for the money invested in the Topila region than in the Panuco region.

Farther south are the large gushers which have shown not only enormous daily capacity but also staying qualities, and are unlike any other wells in the world of very large size. Those are the wells at Casiano and Potrero del Llano.

As regards the geology, a few of these regions have been developed by geological study; the rest have been found more or less by accident. In the Panuco region there are only slight evidences of anticlinal structure, by which a rational development of the oil territory might be planned. When it comes to Topila, after all is said, nothing is known about the geology of it or the relation of the water stratum to that of the oil below. When that is known, very probably the oil can be successfully taken out without running into the water. It will require much more geological study than anybody has yet put on it.

It is interesting to know that in the Potrero del Llano region the great No. 4 well of the Aguila Company was really discovered by careful geological study, which showed a structural dome in a topographic basin. A number of small wells yielded the geological information and enabled Dr. Hayes to develop the big well strictly from geological data.

Philip W. Henry, New York, N. Y.—Mr. Ordoñez's paper brings to mind my experience in that part of Mexico 10 years ago—prior to its development as an oil field—in making an investigation of the residue from oil seepages, such residue being known locally as *chapapote* and

in the paving trade as "natural asphalt." Up to that time the only natural asphalts which had been developed commercially were the two deposits, situated 100 miles apart, known as the Pitch Lake in the island of Trinidad, B. W. I., and the Bermudez Lake in Venezuela. Both of these "lakes" are composed of the residue of asphaltic petroleums, the lighter oils having been distilled by the sun and wind. In the Pitch Lake of Trinidad the petroleum has filled the crater of an extinct mud volcano having a surface area of 110 acres, and in the Bermudez Lake it has spread over the surface of a swamp to the extent of 1,100 acres. From borings which I made in the Pitch Lake in 1894 the depth is at least 150 ft. in the center, which was the greatest depth attainable with the boring apparatus at hand. A boring made in the lake several hundred feet from the edge showed a bituminous sand at a depth of 90 ft., indicating a bowl-shaped crater. The surplus seepage or asphalt has run down the sides of the crater—the lowest edge of which is 110 ft. above sea level—into the Gulf of Paria about ¾ mile distant. Owing to the large amount of asphalt taken from the lake during the past 25 years, from 100,000 to 200,000 tons per year, the present level of the lake, which settles uniformly, but slowly, as a whole, is now some 10 or 15 ft. below the lowest edge of the crater. Seepage is still in progress, but probably at a rate of not more than 5,000 tons per year. The depth of the Bermudez Lake is about 10 ft., and seepage there also still continues. At the present rate of consumption both of these lakes or deposits are practically inexhaustible. In the Pitch Lake of Trinidad the petroleum has become so intimately mixed with water and earth that the residue or crude asphalt has a remarkably uniform composition of 40 per cent. bitumen, 28 per cent. water, and 32 per cent. fine sand and clay, finer even than Portland cement. In the Bermudez Lake the residue or crude asphalt has taken up a certain amount of surface water and earth, varying from 5 to 40 per cent. of water and 2 to 5 per cent. of earth. As refining consists merely in driving off the water, Trinidad refined asphalt contains 55 per cent. of bitumen and Bermudez asphalt about 95 per cent.

These two deposits were originally developed by separate business interests, which made a combination in 1894, thus forming a monopoly in the supply of natural asphalt. This caused a search for other natural asphalt deposits, or oil seepages, among which were those in Mexico lying between the Panuco and Tuxpam rivers, 120 miles apart. This district during the past four years has developed into the greatest oil field of the world. In 1898 to 1900 asphalt was taken from a deposit known as Chapapote Nuñez, some 2 miles north of the Tuxpam River, and from Cerro Viejo, 8 miles north, and used in paving certain streets in Chicago and Jersey City. This asphalt was dug from the deposits, placed in boxes holding about 250 lb. each, floated 60 miles down the Tuxpam river and then loaded on steamers lying outside the bar. The

amount thus shipped was small, probably not over 2,000 tons in all, and the expense was prohibitive, both on account of the small amount of asphalt in the deposit and the cost of handling and transportation. In 1902, A. H. Carner, who had originally developed the Bermudez asphalt deposit, but who at that time was no longer connected with it, became interested with A. L. Barber in these deposits or seepages in Mexico, and obtained oil and asphalt leases to the extent of 80,000 acres scattered between San Geronimo on the Tamiahua lagoon, 60 miles south of Tampico, and Cerro Azul, 30 miles to the southwest. In order to rid the crude asphalt (petroleum residue) of water and earth, he erected a small refinery at San Geronimo, within half a mile of the famous (later) Dos Bocas well, and produced a refined asphalt running about 95 per cent. of bitumen, quite similar to Bermudez. The seepages near San Geronimo being soon exhausted, he brought *chapapote* or crude asphalt from the Juan Casiano lease, 10 miles distant. In all, it is not likely that more than 1,000 tons of asphalt were shipped from San Geronimo, most of it being used in the City of Mexico. In the fall of 1904 I was engaged to make a careful determination of the amount of asphalt on these leases and the cost of building a railway to connect the largest deposits with the Tamiahua lagoon. A cursory examination soon showed that the only deposits of any magnitude were those at Cerro Azul, which could be connected with the lagoon by building a railroad 20 miles in length to Cuchillos on the Tancochin river, and a survey of this route was made.

At Cerro Azul, the crude petroleum, instead of coming up through a mud volcano as at Trinidad or spreading over a swamp as at Bermudez, ran down water courses, and the seepage was so small that, although still going on, there had accumulated during the centuries not more than a depth of 3 ft. of residue or asphalt in any one place. In fact, the seepage was so much less than at Trinidad or Bermudez that the total accumulation in this district was trifling in comparison. To determine the exact amount of asphalt, I took cross-sections every 50 ft., making soundings every 10 or 20 ft. on the section by means of an axe or machete, thus determining the amount in each deposit. Within a radius of a mile I found 14 separate deposits worthy of examination, and the accompanying table shows the average length, breadth, and depth of each deposit, together with the area, volume, and tonnage of asphalt, figuring 70 lb. per cubic foot. As this amount of crude asphalt would not produce more than a total of 20,000 tons of refined asphalt, it was evident that as compared with the Trinidad and Bermudez deposits, which were practically inexhaustible, supplying 150,000 and 30,000 tons respectively per year to the trade, these deposits in Mexico were negligible as far as the business of the world was concerned.

For use in Mexico, however, with its heavy duty on foreign asphalts,

Deposit	Length, Feet	Average Breadth, Feet	Average Depth, Feet	Area, Square Feet	Volume, Cubic Feet	Tons
A	1,900	87.6	1.07	166,525	177,999	6,230
B	400	172.9	0.44	69,175	30,506	1,068
C	800	32.6	0.73	26,100	19,186	672
D	350	107.9	0.28	37,750	10,451	366
E	200	64.5	0.32	12,900	4,106	144
F	1,470	13.8	0.38	20,300	7,600	266
G	1,050	47.1	0.44	49,450	22,030	771
H	1,800	158.1	0.76	284,650	216,006	7,560
I	600	121.2	0.56	72,730	40,116	1,404
J	1,450	93.1	0.94	135,025	126,855	4,440
K	350	75.1	0.72	26,275	18,946	663
L	500	79.8	0.53	39,400	20,962	734
M	1,200	31.6	0.75	37,880	28,868	1,011
N	440	95.5	0.65	42,010	27,117	949
Total....	12,510	81.5	0.73	1,020,170	750,768	26,278

there was a market, which in connection with the development of oil and of agriculture would have warranted the construction of the railroad. Just then, however (1905), Mr. Barber and Mr. Carner again took control of the Bermudez deposit, and as their interest was in asphalt rather than in oil, in 1906 they sold their leases in Mexico to E. L. Doheny and associates, and these properties are now among the most valuable belonging to the Huasteca Petroleum Co. A narrow-gauge railroad 30 miles in length now connects these properties, extending from Cerro Azul to San Geronimo. On the Juan Casiano lease is the famous well, which during the past four years has produced some 34,000,000 bbl. of oil, and is still producing at the rate of 23,000 bbl. per day. Nothing, however, has been done with these deposits of natural asphalt, for as Mexican crude petroleum, like California crude, has an asphaltic base, there can be manufactured out of it an excellent paving asphalt, which is now being used in such important cities as New York, Philadelphia, and Baltimore. During the present year asphalt manufactured from Mexican crude petroleum has been used in paving Fifth Avenue, New York City, between 26th and 34th Streets, thus entering into serious competition with Trinidad and Bermudez natural asphalts. In the development of the oil fields of Mexico, more particularly the district between the Panuco and Tuxpam rivers, it is interesting to remember that it was the oil seepage (natural asphalt) rather than the oil itself which attracted certain capital to this region eight years before its remarkable value as an oil field had been demonstrated.

I. C. White, Morgantown, W. Va.—On my first visit to the Mexican oil fields, in 1911, I was accompanied by Señor Ordoñez, who was the

first one to predict the presence of great oil deposits in Mexico and whose intimate acquaintance with Mexican geology assisted me greatly in obtaining a clear understanding of the vast oil fields of our sister republic. It may not be generally known that the most productive oil well ever yet drilled in the world's history is located in Mexico, and known as Casiano No. 7, situated on the coastal plain about 75 miles south of Tampico, 25 miles west from the Gulf, and probably 20 miles from the famous Dos Bocas gusher. The boring is at the foot of a volcanic hill, 250 to 300 ft. in height, and near where a large seepage or petroleum spring had been issuing through cracks or crevices in the rocks made by the welling up of these geologically recent necks and bosses of diabasic lavas, thus creating the dome-like structure in the underlying Tamosopa limestones, the mother rock of the oil, so favorable to the accumulation of oil and gas deposits in commercial quantity. This well, which is owned by the Huasteca Petroleum Co., a subsidiary of the Mexican Petroleum Co., was struck on Sept. 11, 1910, and up to Sept. 11, 1914, when it was just 4 years old, has produced, according to a telegram just received from E. L. Doheny, President of the company, 33,580,000 bbl., and is still yielding, at the rate of 20,000 bbl. daily, oil of 22° gravity Baumé without any trace of water, while at the same time emitting daily about 10,000,000 to 12,000,000 cu. ft. of natural gas. When one considers the space necessary to hold such an enormous mass of liquid (196,250,000 cu. ft.) and 100,000,000,000 cu. ft. of natural gas, even when under the original rock pressure of 600 lb. to the square inch, it is realized that the total space required to hold this amount of oil and gas would total nearly a cubic mile, and that hence the conditions of the rock reservoir or oil pools in the Mexican fields must be completely dissimilar to those anywhere in North America, or anywhere else in the world for that matter. In all other oil fields of the world, so far as known, the deposits of petroleum and natural gas are stored principally in the pores and minute spaces of coarse sand and gravel or crystalline dolomite without the intervention of any large cavities, but in the case of these Mexican wells that have produced such enormous quantities of oil, as Casiano No. 7, Potrero del Llano No. 4 and others, it is unbelievable that vast open spaces do not exist in the underground limestone reservoirs of these great Mexican gushers. Of course the upward bulging of diabasic necks, bosses, and sills of lava would open large fissures in the limestone, and the accompanying acidulated waters would doubtless form additional solution cavities, the evidence of which Dr. C. W. Hayes, of the Mexican Eagle Co., thinks he has seen in specimens of limestone thrown out of his company's great Potrero del Llano well No. 4, located about 30 miles west of Casiano No. 7. This famous well of the Mexican Eagle Co. is probably the largest oil well ever discovered, since reliable estimates of its production when first struck place the output at 150,000 to 160,000 bbl. daily, all of which

passed down the Buena Vista and Tuxpam rivers into the Gulf of Mexico for a period of 54 days, at the end of which time the flow was turned into an earthen reservoir 20 to 30 ft. in depth and covering an area of 65 acres. It then by actual gauge registered a production of 100,000 bbl. daily. This great well was finally gotten under control, and successfully shut in, although the rock pressure was 850 lb. to the square inch. This was done in 1911, the well having been drilled unexpectedly into the oil horizon about Christmas, 1910. For the last two or three years, it has been permitted to produce 25,000 to 30,000 bbl. daily, or as much as the pipe lines could transport to the loading stations on Lake Tamiahua, and also to the Gulf of Mexico near Tuxpam, where the oil is loaded by a combination of pipe line and rubber hose directly into ocean steamers in the open Gulf. Reports from Mexico state that this famous well was entirely closed in during the recent revolutionary troubles in the Tuxpam region, and that within the last few weeks the oil escaping in large quantity from seepages through fissures in the vicinity of the well, had been set on fire, possibly by lightning. The well is therefore in great danger from the intense heat, which has already destroyed the packing around the joints of pipe lines leading directly from the well, thus permitting a vast amount of leakage of oil, which was also burning. The well itself is securely shut in under a large mound of reinforced concrete and when I saw it in April, 1913, there were no indications of extensive seepages of oil anywhere near the well.

When Casiano No. 7 was struck, it was shut in entirely after it had flowed for a few hours at the rate of 50,000 bbl. daily. When the oil began to spout out of the ground over three or four acres around the well at the rate of 4,000 or 5,000 bbl. daily, the General Manager, Mr. Wiley, opened the valve which closed the well in, until the rock pressure was reduced to only 290 lb. to the square inch instead of 600, the rock pressure of the well when entirely closed in. The oil then ceased to spout through the ground, but went through the pipe line into a wet-weather pond on the borders of Lake Tamiahua until adequate tankage could be rushed to completion. It is probable that the pressure in the Potrero del Llano well could not thus be relieved on account of lack of storage reservoirs during the revolutionary troubles, although at present, in spite of the great fire raging near the well from the leaking lead lines and seepages, other lead lines not affected by the fire are reported to be delivering 30,000 to 35,000 bbl. of oil into tankage to be pumped away to tidewater.

With such wells as these two monsters, and the area of possible oil territory in Mexico only as yet scratched, as it were, it is almost certain that in productive capacity of low-grade (22° gravity Baumé) fuel oil, our sister republic is destined to surpass California and all other known fuel-oil regions of the world.

Gas and Oil Wells through Coal Seams

A discussion at the Pittsburgh meeting, October, 1914.

GEORGE S. RICE, Pittsburgh, Pa.—Undoubtedly there is a serious problem through the juxtaposition of gas and oil wells and coal mines, not only at the present time, but possibly of far more serious import for the future. If one examines a map of the coal fields on which is superposed the oil and gas fields of the country, one finds that they overlap to a very great extent. I may say at the beginning that we have not, so far as I know, had any very large number of accidents growing out of this situation causing loss of life. Such accidents, or rather incidents, that have occurred have all been rather potential possibilities for disasters due to the leakage of gas from wells into the adjacent coal mines. The condition has been most acute in the Fairmont district of West Virginia, and in the general coal fields southwest of Pittsburgh in Pennsylvania. The accidents which have actually occurred were summarized in *Bulletin No.* 65 (Oil and Gas Wells through Workable Coal Beds) issued by the U. S. Bureau of Mines, reporting the transactions of several meetings in Pittsburgh, February, 1913, of oil and gas well representives, State geologists, State mine inspectors, coal-mine operators, and representatives of the U. S. Bureau of Mines. This conference was partly brought about through a case in which the Pennsylvania State inspection department had brought suit to restrain a gas company from completing a well through a certain mine. The office of the Attorney General of Pennsylvania asked the opinion of the Bureau of Mines upon the matter; about the same time the Association of State Geologists, at its New Haven meeting, asked the Bureau to take up the problem. Accordingly, it was thought advisable to bring together the several interests. For the most part the danger is, I think, greater in the future than at the present time. Many parts of the coal fields of Pennsylvania, in Greene County particularly, have been penetrated by numerous wells very close together, and with the present practice of having a large coal pillar around each well, has made or will make the mining of many areas difficult or prohibitive. There is not time to discuss fully what should be done, but it is believed there should be a standardization of the methods.

W. E. FOHL, Pittsburgh, Pa.—I would like to remark in connection with Mr. Rice's preliminary statement that in all these accidents, the accounts of which were collected in the bulletin issued by the Bureau of

Mines, not one of them occurred without the presence of an open light. The gas which leaked into the mine from some place or other, in some way or other, always came in contact with an open flame, and it seems to me that that has a highly practical bearing upon the methods of operation of our mines. To put it in another way, it seems reasonable to suppose that if mines, in which these leakages occurred, had been worked with safety lamps, such as are used elsewhere on account of the presence of marsh gas in the coal itself, none of these accidents would have occurred.

Protecting Pillars.—It is generally conceded that oil and gas wells penetrating workable coal seams require protection from the mining operations in their vicinity, although there have been numerous instances in the McDonald field where, through lack of knowledge of well locations, all the coal has been removed from around the wells without damage to them or the miners engaged in removal.

Various proposals have been made with reference to this protection, ranging from the removal of all the coal, substituting an artificial pillar, to the leaving in place of a pillar of coal 200 ft. square, containing 0.92 acre, which in the Pittsburgh coal seam in this vicinity, having a thickness of 5 ft., would mean the irreparable loss of nearly 7,000 tons of coal for the protection of a single well.

I think it extremely doubtful that any effectual means of replacing the coal *in situ* can be found, and that it provides the best protection against the admission into the mines of deleterious gases in large volumes. It does not appear either necessary or advisable that pillars should be provided that will be absolutely proof against rupture when the final subsidence of the strata takes place after the removal of the surrounding coal. The requirement is that they shall keep the well intact during the preliminary breaks, and it must be borne in mind that these take place from the bottom upward in such manner that, when the heaviest pressure is exerted against the well and its surrounding pillar, actual mining with its accompaniment of human labor can only be in progress at a considerable distance from the well, this distance varying with the thickness of the overlying strata.

I believe there is a safe size of pillar, well within the limits of the maximum size I have quoted, for each of the mining regions in which coal, oil, and gas are being removed simultaneously; but experimental information for fixing the size of this pillar is not yet at hand.

The protection of a wall is analogous to the protection of a shaft, except that protection for the latter ordinarily comprehends support for the more important units of the surface plant. The size of pillar actually requisite for these purposes is not susceptible of mathematical calculation, as it is impossible to forecast either the volume or the direction of the forces which will impinge on the pillar during the subsidence of the overlying strata. These can only be estimated, and the

following table will show how widely authorities on mining have varied in the estimates they have made in this connection. It will be readily seen that there is no consensus of opinion among the seven authorities quoted as regards any given depth and it will also be apparent that there is no regularity of disagreement in progressing from the minimum to the maximum depths to which their formulæ have been applied in the preparation of the table.

Sizes of Shaft Pillars in Accordance with Formulæ of Various Mining Authorities for Different Depths from Surface

Depth from Surface	Feet 100	Feet 200	Feet 300	Feet 400	Feet 500	Feet 600	Feet 700	Feet 800
	Length of side of square pillar							
Merivale	38	54	66	76	85	93	101	118
Andre	105	105	105	105	121	135	154	175
Wardle	120	120	120	130	156	180	204	231
Pamely	120	120	120	145	169	195	220	244
Foster	159	224	275	317	355	389	420	449
Dron	67	133	200	267	333	400	467	533
Hughes	100	200	300	400	500	600	700	800

In 1910, acting in conjunction with the late James Blick, sometime Inspector of Mines for the State of Pennsylvania, the writer advised the leaving of pillars for the protection of oil and gas wells as shown in the subsequent table. This table was intended for use in the mining of the Pittsburgh seam of coal in West Virginia where it had an average thickness of 7 ft. and where the extreme depth of cover was 700 ft.

Pillar Sizes Suggested by Blick and Fohl

Cover, Feet	Size of Pillar Feet Square	Area, Acres	Tonnage Recoverable	Cover, Feet	Size of Pillar Feet Square	Area, Acres	Tonnage Recoverable
20	40	0.04	400	300	120	0.33	3,300
30	50	0.06	600	350	125	0.36	3,600
40	60	0.08	800	400	130	0.39	3,900
50	70	0.11	1,100	450	135	0.42	4,200
60	80	0.15	1,500	500	140	0.45	4,500
100	90	0.19	1,900	550	145	0.48	4,800
150	100	0.23	2,300	600	155	0.55	5,500
200	110	0.28	2,800	650	165	0.62	6,200
250	115	0.30	3,000	700	175	0.70	7,000

Although this table was constructed only after a careful examination of the mining conditions in the region where it was intended to be applied and after a careful consideration of all the available literature on shaft

pillars, it was recommended only tentatively and was accompanied by the following suggestion:

"We have in mind as a suggestion that two wells might be drilled and cased to the coal. One of these to be located near a barrier pillar, so that after the coal had all been removed up to and surrounding three sides of its supporting pillar, access could be had from the remaining side, making it possible to drive narrow headings through it to observe the crushing effect of the overlying strata. Another with its supporting pillar might be left as an island of coal in a considerable area of gob and observations taken as to the effect of removing the surrounding coal, both on the surface and on the casing of the well. The most important information, however, could be secured by a careful watch and instrumental location of all surface breaks at all of the mines of the ——— Coal Company, during the same time, with a simultaneous location of the position of the inside workings at the time breaks were made. From these it should be possible to determine to a nicety the extreme angle at which the strata may be expected to break over the solid coal and from this the size of pillar necessary could easily be established."

So far as the writer has knowledge, none of the observations here named has been made either in the West Virginia region for which they were suggested, or elsewhere; but if, as he believes, a pillar of coal in place is a necessity, it is very certain that its size can be determined only by practical experiments along the lines here suggested.

M. B. Layton, Pittsburgh, Pa.—During the past few years there has been considerable discussion among coal, oil, and gas operators as to the method of drilling through coal measures, having in mind the conservation of life and property. Recently there was held in this city a conference between the coal, oil, and gas operators and the U. S. Bureau of Mines. As a result of these meetings a committee was appointed to prepare and present a bill to the State Legislature of Pennsylvania, covering what was agreed upon at this conference as the best method of drilling oil and gas wells through coal measures. The speaker was a member of this committee, and after considerable labor a bill was drafted and presented to the State Legislature, which passed first reading, and then passed into the archives of bills which never again see the light of day. Why? None of the committee knows. This was a great disappointment to the committee, as the bill as drafted by the coal, oil, and gas men was considered a safe and sane method of drilling through coal measures, having in mind the conservation of life and property. The oil and gas operators are very anxious to remove as far as possible anything that would result in the loss of life or property.

In discussing the drilling of gas and oil wells through coal measures the speaker believes it advisable to discuss the subject in the following manner: Location of well; diameter of hole and casing; the manner of protecting casing through the coal seams; and the plugging of abandoned wells.

The location of a proposed well should be determined by a survey,

showing the courses and distances from two permanent points on the boundary of land upon which the proposed well is located, giving the name of adjoining tracts, township, and county. If the tract is in a locality which is being operated for coal, a copy of the description of plat should be filed with the coal operator. Should the proposed location be such that the bore hole would pass through an entry or room in the mine, the same should be re-located so as to pass through the solid coal except in case of an abandoned mine, providing it does not pass through an entry which is in use for free air to another part of the mine in operation, or a passageway for men to get to and from their work.

Where the coal is in place a hole of a diameter 4 in. greater than the inside diameter of the outside casing to be put through the coal shall be drilled at least 30 ft. below the bottom of the coal bed. Within this hole shall be placed the casing, the space between the outside of the casing and the wall of the hole to be filled with cement mortar to a height of 30 ft. above the top of the coal bed for the purpose of excluding water from the coal bed.

Where the coal is removed and the mine excavation is inaccessible, a hole of a diameter sufficiently large to permit the setting of a liner 4 in. larger in diameter than the inside diameter of the casing to be put through the coal shall be drilled at least 30 ft. below the bottom of the coal bed. Within this hole shall be placed a liner 4 in. larger than the inside diameter of the casing, and extending from the bottom of the hole to at least 30 ft. above the top of the coal bed. A string of casing shall be placed within this liner and the space between the liner and casing filled with cement mortar to the top of the liner. To exclude water, the space between the casing and the wall of the hole and immediately above the liner shall be filled for a distance of 10 ft. with cement mortar.

When the coal is removed and the mine excavation is accessible, a suitable retaining wall is laid in cement mortar about the casing. This wall shall extend from 2 ft. below the mine floor to the roof of the mine, and be of such size as to retain from 4 to 6 in. of cement mortar about the casing. This work to be completed before the well is drilled to a greater depth.

To the speaker, the greatest danger arises in the improper plugging of abandoned wells. There is no part of the work which has been carried on in such a haphazard way. Wells drilled through coal beds, upon abandonment should be filled solidly with rock sediment, sand, clay or other suitable material, from the bottom of the well to a hard and firm stratum below the last string of casing set in above the stratum producing gas or oil. At the top of the lowest producing sand two wooden plugs at least 3 ft. in length should be placed; these to be of size or diameter of the hole and driven into place, and on top of the plugs 10 or 12 ft. of clay should be placed and thoroughly tamped down for the purpose

of preventing the passage of gas, oil, or water. Immediately below the seat of each string of casing there should be driven a seasoned wooden plug, and spaces between wooden plugs should be filled solidly with rock sediment, clay, sand or other suitable material; as the casing is withdrawn, all plugs should be driven in place. In a well where the casing has been cemented in, the outer casing may be cut off at a point not less than 50 ft. above the coal bed and removed, but in any event, the hole should be filled to the surface.

The great difficulty has been a lack of co-operation between the gas and oil operators and the mine operators. *There must be a hearty co-operation and fairness* on the part of each of them if we would conserve life and property. We need a stringent law which would work no injury to either party, and pliable enough to let either party get his product out of the ground at the least possible cost. The procedure suggested does add considerably to the cost of each well, and in following it the gas and oil operator would, as far as we can see at the present time, have done all that he can to safeguard the interest of the mine operator and workmen, and they in turn should be willing to furnish complete details of their mines, so that in the making of locations the well operator would not interfere with the coal operator.

With this in mind, on the part of the gas and oil well operators, we desire to impress on you this idea: That the oil and gas well operators are willing to go to any extent that is reasonable to protect life, and remove from the oil and gas operators the accusations that some have endeavored to place on them—that they are responsible for explosions in mines, which I very seriously question. In drilling through coal measures we frequently find large bodies of gas in the coal, and after an explosion has taken place it is impossible to determine from where the gas comes.

What I have had to say to you is the result of considerable thought on the part of the coal and oil and gas men during the past two years in our meetings in preparing the bill and at Harrisburg.

George S. Rice, Pittsburgh, Pa.—There was one point I would like to hear further discussed by Mr. Layton. If I understood him correctly, he spoke as if it would be unreasonable for the coal operator not to leave a pillar of coal around a well. For the protection of life, where no other adequate means presents itself, a coal pillar should be left around the well, whether paid for by the mine operator or the gas or oil well operator. In my contact with the able representatives of the latter I have found them willing to do everything which they consider essential for safety, but the question is, What is necessary for safety? To solve this question satisfactorily the measures adopted must be acceptable to the mine operator, the oil and gas well operator, the miners whose safety is concerned, and the people as a whole, who are concerned in conserving all the mineral resources, coal, oil, and gas. The safest thing would be not to

mine the coal at all in the vicinity of gas or oil wells, or not to drill the wells in the vicinity of the mines; but this is not a practical solution, as generally the ownership of the coal, the oil and gas, and even the surface, is in separate hands, and each wants to realize on his investment. Hence the need of an orderly, systematic, and safe method of procedure, with a minimum loss of coal while obtaining the gas and oil.

Now, concerning the leaving of pillars, probably the larger the pillar, the safer the condition, but if you have holes drilled only 500 or 600 ft. apart on an average, and leave a pillar of 200 ft. square, or even 100 ft. square, you have very little coal left. If a miner does go on and mine in and around such pillars there will be enormous wastage of coal, because it is not practicable to go in and remove these separate pillars after the gas wells have been exhausted. Therefore, some reasonable compromise has to be made in this matter.

From the standpoint of safety, it is desirable to have the pillars as large as possible, but in the interest of the conservation of resources it is desirable to have them as small as possible. Somewhere between, there must be a compromise.

M. B. Layton.—Mr. Rice seems to get the wrong impression of this idea of drilling through coal measures. When the coal has been taken out, the surface breaks down, and all that the gas and oil well operators want is enough coal or concrete around the casing to protect it from sulphur water. The life of the ordinary gas and oil well is short, and we are certain of the result which will eventually come in the district in which the coal is mined, as to the sinking and breaking down of the strata covering coal measures. If the territory is drilled before the coal is taken out and the casing in the well has been protected as suggested, and in case of abandoned wells properly plugged, there will be no danger of gas escaping into the mine, and you have no right to ask the gas and oil well operator to buy the coal in order to get at the product from below, when he is willing to spend his money in doing everything that is possible for him to do to safeguard and confine the gas in the casing, and in the case of abandoned wells, to the gas-bearing sands or strata.

George S. Rice.—Buying the coal would not meet the proposition of conservation.

M. B. Layton.—Conservation of life is the important thing to us, and I desire to state most emphatically that the greatest danger from gas escaping into coal measures is from the improper plugging of abandoned gas and oil wells.

R. D. Hall, New York, N. Y.—It seems to me that possibly the gas men and the oil men are partly to blame for the difficulties they have with the coal owners because of their desire to drill a great number of

wells. The mining operators could possibly manage to continue mining around gas wells without marked danger if the wells were not so frequent. The coal men could either leave big pillars to protect them, if pillars are desirable, or arrange to take the coal out around them in such manner as to do the least detriment to the wells. But the oil and gas men have drilled holes with such extreme frequency that in certain districts it is almost impossible to make any arrangements at all to safeguard the wells. The gas and oil men seem to have much faith in the strength of the little concreted pipe stems which they are putting down through the earth. If they had had more experience in mining they would realize what terrible catastrophes sometimes occur in mines, and how large an area will sometimes cave in at one time, and they would realize the titanic forces with which their little pipe stems are endeavoring to cope.

A little while ago, in Oklahoma, there was a squeeze which suddenly shut in 13 men. They have never been recovered. The cover of the mine at the deepest point was about 700 ft. It was a huge working with about 15 levels. All of a sudden the mine closed in almost all over. When you are dealing with forces of such magnitude, it is hard to arrange any protection at all for the pipe lines. Pillars 100 ft. through, and even more, have been crushed down with a general movement of the mine. It seems that the oil and gas men ought not to put down so many holes. Indeed, it appears unnecessary that there should be so many.

It also is apparent that the mining authorities ought to regulate their work in such a way that there would be the least danger when the falls take place. There are parts of England to-day where coal is being mined all around the shaft. No shaft pillar is left, the operators depending on the symmetrical falling of the rock to prevent any distortion of the cageway. And as a result of these precautions there is very little of such distortion; in fact, it is doubtful whether just as good results cannot be obtained without a shaft pillar as with one. So with proper arrangements it may be safer to take out all the coal around the bore hole than to leave a pillar, unless that pillar is so immense as to be unquestionably adequate. For it must be remembered that the strain does not cease at the edge of the pillar, but extends well over it. That has been recognized for years, and that action has been said to be due to the "draw." It can be very easily seen how logical that is, because when we build continuous bridges which pass over several abutments we have to put tension members on the top of the bridge over the abutments, or there will be a breakage of the bridge on the top of the abutment, despite the fact that there is a support at that point. And the strain those tension members have to sustain is considerable. So it is true that there is a tensional stress which is important, even where the pillars are extremely wide. It is in providing for this stress of the measures above the coal pillar that we find our perpetual risk. It is questionable to my mind

whether it would not pay the operator to take out all the coal around the pipe, meantime taking precaution to make the workings symmetrical around the bore hole, but that cannot be done as long as the oil and gas men persist in putting down so many holes.

David T. Day, Washington, D. C.—Mr. Hall speaks as though he was laboring under the idea held by some people who are not in the oil and gas business, that they drill gas and oil wells for fun. It is very costly fun, and I doubt whether in these days, particularly in Pennsylvania and West Virginia, any oil or gas hole is drilled on a man's own property, which it is not absolutely necessary to drill. In an anthracite mine gas frequently comes through the coal so rapidly as to make a singing noise. The coal does not hold the gas back at all, apparently. What is the use of leaving any pillar walls? It is of no use in holding back the gas. One inch of cement will hold better than any 10 ft. of coal column. The very important feature was brought out by Mr. Hall, of the chances of the pipe and cement collapsing, unless it is excessively thick, where collapsing of all sorts is taking place in the mine. These are matters which must be given very careful consideration, especially where the chances of deflecting that pipe by roof falls, etc., in the mine are imminent. These may be greater in other mines than they are in the mines of Pennsylvania and West Virginia, but still they must be considered. If Mr. Rice will give us the plans which he has in view for substituting columns of some sort that will keep the pipes from breaking, that will help the subject considerably.

George S. Rice.—There is hardly time to discuss that matter because it involves questions of mining and looks, possibly, a little too much to the future. I am strongly convinced that the hydraulic method of stowing the workings, not filling afterward as practiced in the anthracite district, but as the face advances, in the way now practiced in many European mines, is one that we will come to ultimately in this country. If such hydraulic filling is done, we know, from very positive data that we have, that with the proper filling material we can keep the total subsidence down to 5 to 10 per cent. of the thickness of the excavation, and the overlying strata goes down so evenly that it is not ruptured. I was much interested a couple of years ago in visiting a mine under the famous Krupp works in Germany, where the government had previously forbidden the Krupps to take out the coal, which they owned, under a certain million dollar building because it was manufacturing ordnance; out when this system of hydraulic filling had proved itself so effectual elsewhere they were permitted to mine out all the coal, and it was taken out without any trouble whatever to the building, through the employment of hydraulic stowing. If you can go under a great building of that kind with its foundations and all its elaborate machinery, it certainly

would not be very difficult with comparatively flexible pipe arrangements to take up any small subsidence which might occur. Therefore, I think we can look forward to a time when we can mine right along without regard to drill holes; but that is looking entirely to the future, because it will add to the cost of mining. I cannot see that it can be done at the present time. However, I do think that, even at the present time, in some instances in the course of pulling the mine pillars where the coal bed lies horizontally, it may be possible to remove the pillar around a well by substituting a careful packing of rock and refuse, and arranging the casing with a slip joint at the bottom of the mine workings, the inner casing or tubes carrying the gas being free or unattached to the outer casing. With this arrangement, described[1] at the conference of 1913, the outer casing, which is gripped by the surrounding strata, would be carried down with it gradually as the packing is compressed, without disturbing the inner tubes, which will rest on the strata underlying the coal bed. The packing in such cases would need to be carefully done, with special reference to the general withdrawal of the neighboring pillars, to obtain a regular and vertical subsidence. However, even if there was some side movement higher up, the casing and the free inner tubing would probably be only bent and not ruptured, and, with the maintenance of a free vent to the atmosphere between the outer casing, or casings, and the inner gas tubes, there would seem to be no danger of gas leakage into the mine.

R. D. Hall.—I think the remarks relative to gas and oil wells not being drilled for fun possibly overlooked certain facts. The main reason for drilling so many holes is to make the fields produce large quantities of gas and oil in as short a time as possible, with the idea of producing the greatest profit from each field with the least amount of expense, which is a perfectly natural desire. But on the strength of that anxiety of gas and oil men the coal operators have been obliged to leave pillars and hunt around for pieces not so drilled. Thus their mining work has been delayed and their profits reduced. I may here add that these profits are very much less than the gas and oil men are getting. Another reason why the oil and gas men put down so many holes is that their territory is pretty thoroughly broken up. They do not buy contiguous and continuous territory, and as a result are trying to drain one another's property by their holes. And this method of operation is not really advantageous to the public generally.

O. P. Hood, Pittsburgh, Pa.—It has been my opportunity during the last year to observe the operations in Oklahoma, and see what changes might be desired in the regulations drawn up a year ago. I think I may say that we feel like placing more dependence on mud and less

[1] *Bulletin No.* 65, *U. S. Bureau of Mines*, pp. 15 to 18 (1913).

dependence on cement, wherever it is used. In plugging wells, we are growing a little surer every day that the substitution of clay or mud for wood plugs or cement plugs is the proper thing, and that the protecting of pipe lines by something put between casings, as Mr. Layton suggests, is better done with plain mud than it is with cement.

G. A. Burrell, Pittsburgh, Pa.—Our natural gases usually contain other combustible constituents than methane. The paraffin hydrocarbons, other than methane, amount to about 14 per cent. in natural gas, whereas in normal mine air methane is practically the only combustible constituent, so that by a careful chemical examination of the mine air the presence of natural gas might be detected in time to avert a large inrush.

David T. Day.—It has been stated that no mine accident, due to the gas escaping from gas wells into the mines, has occurred except where there were open lights. I would like to know whether the analyses of the mine air, as far as they have been available in such cases, have indicated a tremendous inrush of gas entirely out of proportion to the proportion of natural gas found in anthracite mines or under normal conditions in gaseous mines, or whether the same condition practically exists in both stages.

G. A. Burrell.—The quantity of methane that comes from some anthracite mines per 24 hr. is equivalent to that coming from some gas wells.

George S. Rice.—I might bring to mind the one especially historic case of the Fairmont mine in West Virginia, where a certain well, which had not been commercially successful, was capped and in some way the gas pressure became equalized through the different casings. Inner casings had been used, but all the casings were covered by a large cap over the outer casing and the openings closed. The pressure was said to have been about 1,000 lb. per square inch. The well passed through a mine (No. 47) but was surrounded by a large pillar, about 125 by 380 ft. in area. There was another mine (No. 49) adjoining, separated from the first mine by a barrier pillar about 75 to 100 ft. in width. On a Saturday or Sunday night when the mines were not in operation and the fans had been shut down, the mines being considered non-gaseous, explosions occurred in each of the two adjoining mines, due to the men going in with open lights and igniting gas. Fortunately there were but few men in the mines, and the mines were very well protected from dust explosions through efficient humidifying, so that the explosions were local and only three men were killed. It was quite freely admitted what the conditions were, and the details have been published in a paper by one of the coal operator's engineers in the *Transactions of the West Virginia Mining Institute* for 1911. It was found after the explosion that there

were a series of gas blowers issuing from the floor for a distance of about 2,300 ft. in a diagonal line extending across these pillars. It is supposed that there may be a line of structural weakness through the strata, and at some deeper point gas leaking from the well under this high pressure had burst up through the floor of the mine. When the cap over the casings was taken off these blowers ceased. Samples were taken from the blowers, the analysis of which showed it to be natural gas, and not mine firedamp or methane. But it is easy to see from this case that if you have a leakage of such a nature, a more serious disaster might result.

David T. Day.—It is hard to conceive of a well being kept closed for commercial reasons, with so high a pressure.

W. E. Fohl.—The well had not been abandoned. It was simply inconvenient to use the gas on account of its being distant from pipe lines. The capping of it was a mistake. It had an outlet, but by mistake of a man in the field that outlet was closed. The pressure, as it was told to me, was 900 lb.

M. B. Layton.—We frequently abandon a well in a field with a rock pressure of several hundred pounds, but the volume of gas is so small that it does not pay to operate the well.

George S. Rice.—While a well may produce too small an amount of gas to be commercially of value, if it is inclosed, or filled in merely at the top, as has sometimes been done, instead of filling from the bottom, it presents a serious menace to a mine. Though the daily flow may be small, or even insignificant, if the pressure is high a large amount may be stored under a pressure of many atmospheres; then if tapped in the mine, it will rush out, and when mixed with 18 times its volume of air makes a great body of explosive gas with tremendous possibilities for danger to life. I call to mind two instances where uncharted, abandoned wells, of which there were no traces on the surface, were encountered in mining; in one of these instances the casing through the coal was still in place when struck by the mining machine; as the casing was not broken, the leakage was not sufficient to be serious. In the other instance the casing had been pulled and the gas came out in volumes and was ignited. Fortunately it was ignited before it became mixed with the air, so it merely burned. While this caused a serious mine fire which involved closing the mine for a period, the drill hole was located by surveying and probing, and was then redrilled, cased from below the coal, and vented to the surface. The mine fire was finally smothered by the erection of underground stoppings; but there was serious loss of output for the operator, and loss of wages for the miners, which would have paid many times over for original complete filling of the well.

Another lesson to be drawn from this is that there should be a law

compelling the public filing of a map to show the exact location of every well, together with a complete record showing the depth of hole, the character and position of all casing left in the hole, and of the plugging or filling. Our State geologists would like even more information about the wells made a matter of record; but there might be some argument raised against giving complete records of the strata, and the gas, oil or other mineral found, but surely there can be no serious objection to the records needed for the safety of future mining, not only in the upper coals or other mineral beds, but also beds at lower horizons not now commercially workable.

George S. Rice.—I think those who were not present at the conference a year ago, realize the amount of work that has been done and the amount of material threshed over before the suggested laws were formulated. It seems to me the greatest danger is from small wells, and those are the ones which frequently have the highest rock pressure. That is the reason it is necessary, in plugging wells, to fill them solid from the bottom to the top, so there is no chamber where gas could accumulate. I think most of the accidents, outside of the one at Fairmont, have occurred in the striking of wells, and the sudden liberation of a small pocket of gas. That is the danger in the old abandoned wells, and that is the object of these specifications for filling wells with soft material from bottom to top.

Another thing that should be mentioned in connection with this is the mud-laden process of drilling wells. I know that certain rather large oil and gas interests think that the drilling of wells by the mud-laden process will eliminate all troubles. Now it is expressly stated by the bulletin on this subject that to utilize the gas which is shut off in this process, you simply pump in the head of water, and the rock is filled with the sediment that is deposited automatically. The sediment in this process is clay, largely colloidal in character. And that condition will be maintained by the extraction of salts from the well walls. I think we can all conceive conditions in which the head of water would not be maintained, and the sand would be automatically opened up.

Chlorides in Oil-Field Waters

Reply to discussion of the paper of C. W. Washburne, *Trans.*, xlviii, 687 to 694.

C. W. Washburne, New York, N. Y. (communication to the Secretary*).—Professor Lane makes an interesting contribution to the study of chloride waters, in saying that calcium chloride waters occur not only in the greenstones of the Lake Superior copper mines, but also in the interbedded felsitic conglomerates and "in far-removed sandstones, *e.g.*, near Whitefish lake, and Ontonagon, in the Michigan iron mines, and in the Storm King granite." Surely these are not connate waters preserved from the Algonkian sea, assuming that the Algonkian sediments are marine rather than continental. Finding the same type of water in the Storm King granite points rather toward an abyssal source. The possible connection of these waters with the magma of the greenstone is immaterial, although some basic calcic magma is the most probable source. It is to be expected that the ascending water of the region would be found not only in the greenstones, but in all the rocks which it could penetrate.

The ascent of water of this kind along vertical passages, and the consequent addition of a common ion to the circulating sedimentary salt solutions, would explain the precipitation and localization of the "giant, concretionary" salt plugs of the Gulf Coast, without making the excessive demand for an improbable quantity of salt in the adjacent sediments that is required by the unmodified theory of Harris. Without such a precipitating medium most of the dissolved sodium chloride would have been lost by passing on to the surface without precipitation. The base to which the chlorine ions were attached is immaterial, providing they were sufficiently concentrated, but the presence of the chlorides of calcium and magnesium in the solutions is indicated by the associated bodies of secondary dolomite.

Volcanic free chlorine or hydrochloric acid could not migrate far without picking up bases, which would be largely magnesium and calcium if the source were in underlying plugs of magnesio-calcic magmas. Rocks of this type reach the surface near by in the peridotite dikes and pipes cutting Cretaceous strata at the inner margin of the coastal belt

* Received Mar. 6, 1915.

in southeastern Arkansas,[1] and in the many dikes and plugs of olivine basalt cutting Cretaceous and Eocene strata in Mexico, along several hundred miles of the coastal belt itself. Where the stratigraphic cover is thicker, and the fracturing apparently less intense, as in Louisiana, coastal Texas, and southern Mexico, it is quite reasonable to believe that the intrusions failed to reach the surface and are deeply buried, manifesting themselves only in the peculiar phenomena of the salt plugs of these terminal regions.

Besides the Arkansas and Mexican intrusions, there are peridotite dikes of unknown age, Triassic or later, cutting Carboniferous strata in Elliott and Crittendon counties, western Kentucky,[2] one of which extends into southern Illinois. Also there are dikes of similar peridotite at Syracuse[3] and at Ithaca,[4] New York. It is suggestive that all of the intrusions in or near the eastern group of American oil fields are of this ultra-basic type. Wagner[5] says that the New York, Kentucky, and Arkansas peridotites are essentially identical with the diamond-bearing kimberlites of South Africa. Sir Henry Roscoe[6] extracted aromatic hydrocarbons from the kimberlite of the Kimberley mine. In the Kentucky peridotite, or kimberlite, as Wagner calls it, Diller recently has found particles of metallic iron suggesting the carbide-bearing iron in a basic phase of similar rock at Ovifak, Greenland. The Arkansas peridotites carry diamonds. All of the Mexican intrusions that I examined, some 20 or more, contained oil. In Cuba there is a striking association of oil with serpentine, including many occurrences. The appeal to magmas of peridotite nature is therefore not far fetched. In many respects these rocks resemble the meteorites in which hydrocarbons have been found, which are supposed to be fragments of disrupted asteroids or other heavenly bodies. The inorganic nature of the meteoric hydrocarbons is, of course, unquestionable.

From these and other arguments it seems that the hypothesis of

[1] J. C. Branner and R. N. Brackett: *American Journal of Science*, 3d ser., vol. xxxviii, No. 223, p. 50 (July, 1889). G. F. Kunz and H. S. Washington: *Trans.*, xxxix, 169 to 176 (1908). A. H. Purdue: *Economic Geology*, vol. iii, No. 6, p. 525 (Aug.–Sept., 1908). Include references to diamonds.

[2] J. S. Diller: *American Journal of Science*, 3d ser., vol. xxxii, No. 188, pp. 121 to 125 (Aug., 1886); *Bulletin* No. 38, *U. S. Geological Survey* (1887); *American Journal of Science*, 3d ser., vol. xliv, No. 262, p. 286 (Oct., 1892).

[3] G. H. Williams: *American Journal of Science*, 3d ser., vol. xxxiv, No. 200, p. 137 (Aug., 1887); *Bulletin of the Geological Society of America*, vol. 1, p. 533 (1889); *Science* (Mar. 11, 1897). C. H. Smyth, Jr.: *American Journal of Science*, 4th ser., vol. xiv, No. 79, p. 26 (July, 1902).

[4] George C. Matson: *Journal of Geology*, vol. xiii, No. 3, p. 264 (Apr.–May, 1905).

[5] P. A. Wagner: *The Diamond Fields of Southern Africa*, p. 106 (Johannesburg, 1914).

[6] *Proceedings of the Manchester Literary and Philosophical Society*, vol. xxiv, p. 5 (1885).

the igneous origin of oil has sufficient elements of probability to be worthy of consideration, at least tentatively. The method of multiple working hypotheses, developed by Chamberlin,[7] is the only scientific way to attack a problem of this kind, in order to be *able to perceive* all of its various significant features. In this way we have the best opportunity of finding conclusive scientific evidence of the origin of oil, which is completely lacking to-day. It is possible, moreover, that the conflicting evidence will ultimately be harmonized by the conclusion that some crude oils are of organic, others of inorganic, origin. The conservative student will reserve judgment until a complete theory is developed and tested in the field and laboratory.

Every one will agree with Hunt, Becker, Lane and others, that the composition of the ocean must have been changed by contributions from the rivers, also that juvenile and volcanic waters may have affected it, especially in pre-Cambrian time. Under the planetesimal hypothesis of Chamberlin,[8] all surface waters were derived originally from the interior of the earth, and the composition of the primitive ocean would be nearly that of the juvenile waters of the time. The calcium chloride waters in the Algonkian and other rocks of the Lake Superior region conceivably may be of this nature, if they have risen from the Archean. Whether they are called "connate-Archean" or "juvenile" is immaterial.

We do not know the original composition of the ocean and there are only hypotheses of the rate and character of the changes it has undergone. However, under any hypothesis, there has not been sufficient time since the Paleozoic to account for the amount of change in sea salts that is required by the theory that, in the deep wells under consideration, the waters are connate with the strata now containing them. Considering only the ratio of Na : Cl, some of the well waters would represent an ocean only one-fourth, others one-half, as old as the present sea. If allowance be made for any sodium in the original sea water they would be correspondingly older. Under the connate-water hypothesis, they should occur, therefore, in the pre-Cambrian rather than in late Paleozoic formations.

The composition of the Paleozoic sea water is largely hypothetical. It has been suggested, on biological grounds, that it must have exerted approximately the same osmotic pressure as to-day in order to have supported the same types of life. If so, the pressure was maintained by greater dissociation, because the Paleozoic sea water must have been more dilute than now. Yet the very deep well waters are concentrated brines, one of them having about seven times the salinity of modern sea water. To apply the connate hypothesis to these waters,

[7] *Journal of Geology*, vol. v, No. 8, pp. 837 to 848 (Nov.–Dec., 1897).

[8] Chamberlin and Salisbury: *Geology*, vol. ii. See also R. T. Chamberlin, The Gases in Rocks, *Carnegie Institution Publication No.* 106, pp. 70 to 75 (1908).

it is necessary to assume that they were concentrated by evaporation, either before or after burial in the sediments. The former would be possible locally, where shallow, semi-detached embayments and small inclosed seas permitted great evaporation, but it could hardly explain the frequency of deep concentrated waters in many places. The latter method, concentration in the rocks, is possible on the assumption that there are rising rock gases, which drive the water out and evaporate that left behind in the deeper rock pores. Neither of these hypotheses has much strength at present, and I have hesitated to suggest the latter, which if carried far would choke the passages with mineral matter. This would explain the cessation of the ascent of water in ore veins, etc., but it is doubtful that the choking of the passages would not occur too early to permit the effective operation of the process. More likely there is also water rising with the gases and keeping the passages open, but in any case it seems likely that these deep ascending waters are strong solutions. The dryness of the deep rocks can be explained possibly better by the process of capillary concentration, described in another paper.[9] After the cessation of water circulation, by the exhaustion of the magmatic or other source, or the closing of communication, the nearly stagnant water may have been transferred to the finest pores and joints by capillary exchange with the rock gases, which would be concentrated in the larger openings. The water would not flow readily from these minute capillaries into mine workings.

The theory of non-migratory connate waters fails especially to account for the wide variation in the ratio of Na : Cl, as well as its great defect below any ratio possible in late Paleozoic times. A modified theory would be more acceptable, by assuming that the deep well waters have risen from different horizons of originally connate waters in the basal Paleozoic and pre-Cambrian rocks, in which the ratio of Na : Cl might be as low as in the well waters. Thus they would not be connate with the strata now containing them, which they would have invaded from below.

The evidence of the analyses consists in an excess of chlorine with reference to sodium which is far above anything possible in stagnant connate waters. The simplest explanation is that there has been a direct contribution of calcium and magnesium chlorides, the salts to which the present excess chlorine must be assigned in calculating the analyses. This would be effected by the ascent across the strata of water resembling that in the ancient rocks of the Lake Superior region, and its mixture with overlying connate waters.

In examining the analyses of Ohio waters I was struck by the number of analyses in which the ratio of Na : Cl is almost the same as in the water of the existing ocean. In some analyses not only this ratio but also the

[9] The Capillary Concentration of Gas and Oil, this volume, p. 829.

percentages of sodium and of chlorine in the total salts are almost the same as in modern sea water. These must be true connate waters and they indicate a surprisingly small subsequent modification in sea water. In examining the few analyses of waters in oil fields in Tertiary strata I failed to find any which did not appear to be greatly contaminated with the local circulating ground water. It seemed unfair and meaningless to use the brines from the salt plugs of the Gulf Coast, although they present some interesting features.

Professor Lane doubtless has studied this question much more thoroughly, and I await with interest his paper on the chemical evolution of the ocean. It will be surprising, however, if he can manage to explain the great and widely varying excess of chlorine in the water of the deep wells of Pennsylvania and Ohio, by the unmodified connate hypothesis. His seas must have changed remarkably in composition from place to place, unless he admits with me that the waters may have risen from different horizons of connate water in deeper strata, or else that there must have been extensive contributions of more deep-seated (juvenile?) waters in calcium and magnesium chlorides. In either case an ascent across the strata is required.

Professor Lane finds "a serious difficulty" in getting the chloride waters across the St. Peter sandstone, because where that formation can be drilled it now contains a comparatively fresh artesian water. However, it should be noted that wells in the oil fields cannot reach the St. Peter sandstone, and that we do not know the character of the water in that formation beneath the oil pools. From our experience in the deep wells of oil fields we may suspect that the water of the St. Peter sandstone beneath the oil fields may not have the same character as in the areas of artesian circulation, but that it contains either a brine or a sulphur water, or more probably that it is dry at such great depths.

An identical case is furnished by the Dakota sandstone of Colorado, which commonly contains an artesian circulation of somewhat alkaline but comparatively fresh sulphato-carbonate water similar to the common ground water of the outcrop region along the front of the Rocky Mountains. Yet the same sandstone contains an entirely different water beneath the Florence oil field; namely, a hot water, highly charged with hydrogen sulphide and with salts, in which I believe the sulphates and chlorides[10] are most conspicuous. This water was encountered in a well drilled near the southeast margin of the field, where the strong solution quickly corroded away the iron casing, and also in the well on the northwest side of the field which now supplies the Canyon City swimming pool. Similar hot sulphur water escapes from springs in the margin of

[10] Written in the field, from memory, without notes: subject to error in the character of the salts.

the granite west of Canyon City, pointing to its source. If we had only the evidence of the artesian wells in the Dakota sandstone of the surrounding region, we would never suspect the existence of this type of water in the same formation beneath the Florence oil field. It is quite possible to make the same mistake in connection with the St. Peter sandstone.

The localized occurrence of the hot sulphur water in the Dakota sandstone of the Florence oil field is probably not without significance, because sulphur waters are generally characteristic of the Rocky Mountain oil fields, especially in Wyoming, and of many other oil-bearing regions; *e.g.*, the Apscheron peninsula, the Gulf of Mexico, the deep sands of California, etc. Sulphur has an association with oil similar to that of chlorine, which at once suggests the fact, discovered by students of ore deposits, that sulphur and chlorine are the principal mineralizers of the basic magmas, as distinguished from fluorine, etc., for the acid magmas. It is to ultra-basic magmas that the supporters of the volcanic hypothesis must look for the source of oil. The true reason for the association may be far different from the one suggested, but that it is a real association, not without some significance, cannot be doubted by any one familiar with the phenomena of the different oil fields of the world, in all of which, so far as I know, one or both of these substances, sulphur and chlorine, are conspicuous.

I would like to ask the students of ore deposits how they explain the fact that the ground water appears to have a lower limit. The joints cannot be tightly closed at the depths attained by mines. Why has the water ceased to flow at these depths where it formerly deposited minerals? Why should it be so concentrated at some deep places, as in the lower levels of Lake Superior copper mines? Do you think the water could have been shoved out by ascending rock gases? Have you any pressure records, or analyses, of deep-mine gases that suggest an outward creep, or diffusion, of abyssal gas, such as that furnished by the helium of the Kansas natural gas? Can you apply the process of capillary concentration to explain the dryness of the deep fissures? It seems to me that you have some problems in common with those who are studying oil fields, and that it would be profitable for us to get together with our various lines of evidence and our different methods.

POSTSCRIPT.*—I have contended that the composition of certain deep underground waters indicates that waters rich in chlorine have ascended across the strata, from lower sources. Professor Lane argued that the freshness of the water in the St. Peter sandstone is difficult to explain, if highly saline waters have risen across it. To this I replied that the fresh waters characterize areas of artesian circulation, and that we do not know the character of the water in that sandstone beneath

* Received May 20, 1915.

The Origin, Mining and Preparation of Phosphate Rock

BY E. H. SELLARDS, TALLAHASSEE, FLA.

(Pittsburgh Meeting, October, 1914)

PHOSPHATE rock like most other mineral substances is found in nature in varying degrees of purity. Of the impurities that are present some are constituents of the rock itself; others are inclusions of a foreign substance within the rock; while still others represent merely associated materials or minerals, either clinging to the rock or found in cavities and natural depressions, and hence largely removed in mining. Some of these impurities are distinctly deleterious to the processes of manufacture for which the phosphate is mined, while others, although neutral in action or nearly so, yet by their presence reduce the average grade of the rock and thus add useless bulk to the shipment.

It is the object of refined processes of mining to bring the product, as delivered from the mine, to the highest possible grade consistent with the market requirements and demands. This, however, is not accomplished without actual loss in the form of discarded phosphate. It is evident, therefore, that the devising of methods for reducing this loss in mining, and yet maintaining the grade of the rock which the market requires, is one of the serious problems that the producers have to face.

MINERALS OF PHOSPHATE ROCK

The minerals included under the term "phosphate rock" are the calcium phosphates. Of these, apatite is perhaps the most definite and constant in composition, although of this mineral two varieties are recognized, namely, fluorapatite, $Ca_5(PO_4)_3F$, and chlorapatite, $Ca_5(PO_4)_3Cl$. Moreover, the calcium of this mineral may be partly replaced by manganese, forming yet another mineral, manganapatite; or the mineral may become hydrated, forming hydroapatite, which is found as mammillary deposits often not unlike chalcedony in appearance. The term "phosphorite" has been applied to the massive amorphous deposits of phosphate, which may be compact, earthy or concretionary.[1] Among

[1] Dr. Austin F. Rogers, who is investigating phosphate minerals, states that phosphorite or phosphate rock seems to be a mixture of two minerals, amorphous collophanite, largely a solid solution of calcium carbonate in calcium phosphate, and crys-

other varieties of apatite may be mentioned staffelite, which contains a small percentage of both iron and aluminum. It is of interest to note also that this variety is believed to result from the action of carbonated waters on phosphorite, and hence is likely to occur incrusting ordinary phosphate rock acted upon by carbonated waters. Another variety, pseudoapatite, contains both sulphur and carbon dioxide. Of the many other calcium phosphate minerals some closely approximate apatite while others grade into compounds so variable and indefinite in composition as scarcely to be classed as minerals. The deposits of phosphate found in nature evidently contain a number of calcium phosphate minerals, the constituent impurities of which affect the market value of the rock. The aluminum phosphate, wavellite, should also be mentioned since it is mined to some extent as a source of phosphorus.

Associated Minerals

Various other minerals in nature are found associated with the calcium phosphates. This association is sometimes due to actual relationship between the minerals. On the other hand, the association of minerals may be purely accidental, or incidental to the manner of formation of the deposits. With regard to the related minerals, it is apparent that where the calcium phosphates are abundant, other phosphates are likely also to occur. In fact it is scarcely to be expected that extensive calcium phosphate deposits will be found without the presence of at least a limited amount of other phosphate minerals. This is particularly true of iron and aluminum phosphates. These two bases are widely disseminated in nature, and moreover they combine readily with phosphoric acid to form phosphates. Of the iron phosphates, the mineral vivianite, although occurring in relatively small quantities, is widely distributed in nature, and may occur in limited quantities in phosphate deposits, usually as an incrustation, or as an alteration product of other minerals. The iron minerals frequently form in bogs, and it is an observed fact that the phosphate deposits in such localities not infrequently contain more iron than do the same deposits when found on the uplands. In such cases the iron is doubtless a comparatively recent infiltration, and may include phosphates of iron as well as oxides and other iron minerals. Of the aluminum phosphates a large number are known, one of which, wavellite, as already stated, is mined as a source of phosphorus. This mineral and others of the aluminum phosphates are likely to occur in association with calcium phosphate.

Some of the large phosphate deposits have been formed by the replace-

talline dahllite, a calcium carbonophosphate with the formula $3Ca_3(PO_4)_2.CaCO_3$, analogous to fluorapatite. The amorphous collophanite gradually changes to the crystalline dahllite. (Personal letter of May 23, 1914.)

ment of an original rock by calcium phosphate. In this process parts of the original rock not infrequently remain unchanged or incompletely phosphatized. Since the phosphatizing processes proceed from the surface, the imperfectly phosphatized remnant is likely to lie within the rock, thus giving rise to included impurities that are difficult to eliminate. Moreover, small amounts of clay and silica are usually found in the limestone, and as these substances do not readily phosphatize, if not leached out, they remain as impurities in the rock.

Aside from these related minerals, the materials associated with the phosphate rock are varied in character. They include clay, fragments of limestone, flints, gravel, silica in the form of sand, and other resistant materials, the character of which is determined by the manner of formation of the deposits. The associated materials of this nature make up the matrix in which the phosphate rock is imbedded. It is scarcely possible in mining to remove entirely all associated minerals, and the purity of the rock as delivered to the market is affected, without doubt, by the presence of more or less of these minerals, as well as by the constituent impurities of the rock.

Objectionable Impurities

Of the impurities contained in or associated with phosphate rock, the most objectionable in the processes of manufacture of acid phosphate for fertilizers, for which purpose the phosphate rock is almost wholly used, are iron and aluminum. For this reason practically all phosphate mined is sold under a guarantee that the combined iron and aluminum, expressed as oxides, do not exceed a given small percentage of the whole, from 2 to 4 per cent. being allowable. Iron when present in excess of about 2 per cent. brings about reactions which result in the formation of a gelatinous substance injurious to the mechanical condition of the mixture, occasioning also a loss of soluble phosphoric acid. A first step in the reaction with the iron is probably as follows: $2FePO_4 + 3H_2SO_4 = 2H_3PO_4 + Fe_2(SO_4)_3$. Of the sulphate of iron thus formed, a part, according to Fritsch,[2] reacts on acid phosphate of lime, thus forming the objectionable gelatinous precipitate. Owing to the demand of calcium sulphate for water, hydrated iron phosphate, which is a product of these reactions, may subsequently become dehydrated and insoluble, thus causing the loss of available phosphoric acid.

Aluminum existing as a silicate in phosphate rock is likely to be injurious, since according to Fritsch, if not decomposed by the acid, it may cause a part of the phosphoric acid to retrograde. However, when existing in the rock in small amounts as a phosphate, the aluminum is supposed not to occasion a loss of phosphoric acid, both the hydrated

[2] J. Fritsch: *The Manufacture of Chemical Manures*, p. 79 (1911).

and the non-hydrated phosphate being soluble in the precipitated condition in phosphoric acid.

Carbonates of calcium when existing in small quantities in phosphate rock, are beneficial rather than injurious. When the ground rock is treated with acid the carbonate is the first of the ingredients to be attacked, and the heat thus engendered promotes subsequent reactions among the other constituents. Moreover, the carbon-dioxide gas given off from the carbonate lightens the mixture and facilitates drying. Phosphate rock low in or lacking carbonate develops little heat in mixing, and reacts slowly. In such cases this constituent must be added. It is true that the presence of the carbonate necessitates the use of an increased amount of acid, which in turn results in the formation of an increased amount of calcium sulphate or gypsum. The amount of carbonate that is desirable is sometimes given as 5 per cent., but the limits are not strict, and the manufacturers do not as a rule find it necessary to specify directly the amount of the carbonate that the rock must or must not contain. Indirectly, however, an excess of the carbonate is guarded against by other requirements as to the grade of the rock. If it is true as elsewhere stated in this paper that the principal mineral of the massive phosphate deposits is a calcium carbono-phosphate, this fact will afford an explanation of the presence of the desired amount of carbonate in all phosphate deposits of this class.

The fluorine found in phosphate rock, upon being attacked by the acid, forms hydrofluoric acid gas which passes into the atmosphere, it being estimated that as much as from 50 to 66⅔ per cent. of the fluorine present is eliminated in this way. Although a small amount of acid is used up in this reaction and a proportionate amount of calcium sulphate formed, yet it is seldom, if ever, necessary to specify against the fluorine content of the rock, the amount present being negligible.

Among the numerous other impurities that may be present in phosphate rock, silica and clay are perhaps the most common. Here also should be mentioned moisture, which when present not only adds bulk to the shipment but also interferes with the processes of manufacture. The excess of moisture must, therefore, be removed by drying, not more than 3 or 4 per cent. being allowable in the rock as shipped.

The Origin of Phosphate Deposits

The origin of phosphate deposits is such that the presence of associated minerals as well as constituent impurities is almost invariable. The original source of phosphorus, the constituent for which phosphate rock is mined, is in igneous and crystalline rocks, where it exists in combination with other elements forming phosphate minerals. These minerals, as indeed is true of all minerals, are soluble in water, the degree of solubility,

however, varying with the different minerals, and with the diverse conditions to which they are subjected. Indeed, some very interesting and suggestive observations have been made on the relative solubility of phosphates under varying conditions. Thus it has been shown that the solubility of the phosphate minerals is increased by the presence of decaying organic matter in water. They have also been found to be appreciably soluble in carbonated waters. In this connection Reese[3] has made the very important observation that while the phosphate dissolves freely in waters containing decaying organic matter and in carbonated waters, yet when allowed to stand over calcium carbonate, the phosphate is redeposited. In summing up his observations Reese says: "This experiment shows that phosphates may be transported in hard waters, but on standing on calcareous beds would tend to be given up." In speaking of hard waters the author evidently has in mind waters containing, among other things, carbon dioxide in solution, and his conclusion is that such waters will or may drop the calcium phosphate from solution when they stand over limestone. These observations, if true, have two important corollaries, one of which has been noted by Clarke,[4] namely, that in the presence of the carbonate, the phosphate would probably not be dissolved, while the carbonate could pass into solution, thus leaving the enriched residue of phosphate. The second corollary is that calcium phosphate taken into solution by the soil waters at and near the surface may be thrown out of solution in case the water stands for a time at a lower level in the earth on or over limestones. That this process may have been and probably was a factor in the formation of large phosphate deposits resting upon limestone will be shown in the subsequent pages of this paper.

The rain water in passing through the soil and surface materials receives organic acids from the decay of vegetable and animal matter. It also receives carbonic acid which is held in solution, the water thus becoming carbonated, and hence more efficient as a solvent. The original rocks and the soils derived from them contain particles of the phosphate minerals, which when acted upon by the ground waters pass slowly into solution. It is through the solution of the mineral, its removal and subsequent redeposition that workable phosphate deposits are formed. When it is remembered that the phosphorus in the igneous rocks amounted to merely a fractional part of 1 per cent. of the whole,[5] and was without doubt originally widely disseminated, the importance of the processes of concentration of the mineral by ground water and the extent to which they have operated becomes evident.

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[3] *American Journal of Science*, 3d series, vol. xliii, p. 402 (1892).

[4] *Bulletin No.* 330, *U. S. Geological Survey*, p. 443 (1908).

[5] According to Clarke, *Bulletin* 330, *U. S. Geological Survey*, p. 32 (1908), the amount of phosphorus in the lithosphere is 0.11 per cent.

The removal of the mineral from the original rocks and its concentration in later rocks is by no means a simple process. There are as a rule many intermediate stages, the load being taken up, dropped again for a time, only to be once more started on its journey. Among the primary results of concentration and enrichment may be mentioned the phosphate deposits of the crystalline rocks, where the mineral is found in veins, being more or less perfectly crystallized as the mineral apatite. Of deposits of this class some, including those of Canada, are of economic importance, and would be more extensively worked were it not that other and more cheaply mined phosphate deposits are available.

Of the phosphate taken into solution by the ground water a part is taken up from the soils through the roots of plants, and thus becomes a constituent of the plant life of the earth. From the plants the phosphorus passes to plant-eating animals, and through them to carnivorous animals. Phosphorus thus becomes a constituent of the organic life of the earth. The bones of the vertebrate animals in particular contain an appreciable amount of calcium phosphate. It seems well established also that certain of the important phosphate deposits, as well as the guano deposits, are derived from excrement and remains of gregarious animals, particularly birds. It is also true that a part of the phosphate taken into solution by the ground waters is again thrown out owing to changed chemical conditions, and in this way important phosphate deposits are formed. In either case, however, the phosphate may be regarded as only temporarily delayed in its round of circulation. Ultimately phosphate is carried in solution in the ground waters through springs and rivers to the ocean. While the amount in solution at any one time is relatively small, yet, through the continued operation of this agency over long periods of time, a large amount has reached the ocean.

The phosphate carried into the ocean is again removed from solution through the agency of organic life, or, owing to changed chemical conditions, is precipitated. Of the animals that utilize phosphorus taken from the sea water in the construction of a shell covering or skeleton, the best known perhaps is the brachiopod, *Lingula*, the shell of a recent species of which has been found to contain 85.79 per cent. of calcium phosphate. The tests of the crustacea, although less distinctly phosphatic than the shell of *Lingula*, contain an appreciable amount of phosphate. Thus the shell of a recent lobster was found to contain 3.26 per cent. of calcium phosphate, while the lobster as a whole contained 0.76 per cent.[6] The aquatic plants also utilize some of the phosphorus in solution in the water and through them the phosphorus passes to the skeleton of vegetable-eating aquatic animals, and through them in turn to the carnivorous animals.

[6] *Bulletin* 330, *U. S. Geological Survey*, p. 448 (1908).

The phosphorus taken from solution by chemical action is evidently considerable, since nodules of chemical origin, high in phosphates, are found somewhat abundantly in the bed of the ocean. Some of these nodules, reported by Clarke,[7] contain 19.96 to 23.54 per cent. of phosphoric acid.

The amount of phosphate that finds its way into the sedimentary formations through organic and chemical agencies is thus undoubtedly considerable, resulting in the enrichment of certain deposits, which, if not workable, at least serve as an important source of phosphate from which by further concentration workable phosphate deposits are formed.

In this respect the deposition of calcium phosphate is analogous to that of the related mineral calcium carbonate, although of the carbonate much more extensive deposits accumulate than of the phosphate.

Silica (SiO_2) in its round of circulation in the earth presents some interesting analogies and yet strong contrasts to both calcium phosphate and calcium carbonate.

While the round of circulation of phosphate minerals is capable of demonstration as a normal process comparable to that of other common minerals, yet the actual processes of the accumulation of large workable deposits of phosphate rock are in many ways complicated.

Florida Phosphate Deposits

The complexity of origin of the phosphate rock and the manner in which impurities are included in the formation is well illustrated by the Florida phosphate deposits. Of these there are two distinct types, known respectively as the hard-rock and the land-pebble phosphates. These differ materially in their location, origin, and manner of occurrence. The hard-rock phosphates lie in a belt along the Gulf side of the peninsula, extending in a general north and south direction roughly paralleling the Gulf coast for a distance of about 100 miles. The land-pebble phosphate deposits are found farther south, lying chiefly in Polk and Hillsboro counties.

The hard-rock phosphate deposits rest upon a thick and very pure, light-colored, porous and cavernous limestone known as the Vicksburg formation, which is of Lower Oligocene age. At present, in that section of the State no formation other than the phosphate itself lies on top of this limestone. It has, however, been demonstrated by the combined observations of several geologists that certain formations of which only a residue remains formerly extended across the area that now holds the hard-rock phosphate deposits. The formations referred to are Chattahoochee limestone and Alum Bluff sands, both of which are of Upper Oligocene age. These formations are now found bordering the hard-rock phosphate area. The proof of their former extent has been given else-

[7] *Bulletin* 330, *U. S. Geological Survey*, p. 105 (1908).

where and need not be repeated at this time.[8] The hard-rock phosphate deposits are made up largely of the residue of these formations which have disintegrated *in situ*, and accordingly consist of a mixture of materials of the most diverse character including sands, clays, limestone fragments, pebbles and water-worn flints, vertebrate, invertebrate and plant fossils; in fact a heterogeneous mixture of the relatively insoluble and resistant elements of the earlier formations.

The phosphate itself is derived from the Alum Bluff sands, the later and thicker of the two formations that have disintegrated. This formation, the Alum Bluff, in some places reaches a thickness of several hundred feet, and has a large areal extent reaching from west Florida, through

FIG. 1.—MINING PHOSPHATE ROCK BY DREDGE IN FLORIDA.

northern and central Florida, into southern Florida. Throughout its entire thickness, and throughout its whole areal extent this formation is distinctly phosphatic, although in no instance is the phosphate in this formation sufficiently concentrated to form workable deposits.

While these formations, the Chattahoochee and the Alum Bluff, were disintegrating in the area that is now the hard-rock phosphate region,

[8] Origin of the Hard Rock Phosphate Deposits of Florida, by E. H. Sellards, *Fifth Annual Report, Florida State Geological Survey*, pp. 23 to 80 (1913). Recent investigations have shown that the Alachua clays represent a local phase of the hard-rock phosphate-bearing deposits, described in this report as the Dunnellon formation.

the calcium phosphate from the Alum Bluff formation was gradually being taken into solution by ground water and was being redeposited at a lower level in the earth, thus forming the workable hard-rock phosphate deposits. In this process the replacement of the original limestone by calcium phosphate was an important factor, and these deposits afford excellent illustration of the formation of phosphate rock by the replacement process, the shells of the original limestone in many instances retaining their form, although changed chemically to calcium phosphate. In addition to replacement, other processes are observed, prominent among which is the formation of the phosphate by precipitation from solution in a manner similar to the formation of calcium carbonate deposits in caves. This process is evidently secondary, and being now operative is to be

FIG. 2.—REMOVING OVERBURDEN FROM PHOSPHATE ROCK IN FLORIDA BY HYDRAULICKING.

observed in the phosphate boulders themselves, in which all existing cavities are being gradually filled by the accumulation of calcium phosphate. By this process pinnacles are formed hanging from the roof of the cavities, while successive layers of phosphate are spread out over the floor of the cavities. This method of the formation of phosphate deposits has given rise to very high-grade phosphate rock, the Florida hard rock grading, under present methods of mining, 77 to 80 per cent. tricalcium phosphate, while individual specimens contain 84 to 85 per cent.

While the origin of the hard-rock phosphate in its present form is thus clearly evident, there yet remains a large field of investigation to determine the chemical processes by which the phosphate is first taken into

solution, and is subsequently redeposited. Some of these processes, however, are well understood. That normal phosphate is to some extent soluble in soil waters is well established and fully recognized. In the hard-rock phosphate section of Florida there is practically no surface drainage, the rain water passing directly into the earth. At a lower level the circulation of the water is interfered with and the water may become stationary or nearly so. The check in circulation is due in some instances to masses and beds of clay residual from disintegrating formations. In any case the movement of the water is checked upon reaching the water line. The relation of the phosphate deposits to the ground-water level and also the evident and probable changes of the water level during geologic time have been discussed in the writer's paper on these deposits previously referred to. While the processes that brought about the deposition of the phosphate may not be fully understood, yet it is apparent that there are important changes in the chemical conditions. Among these may be mentioned the check to the free movement of the water, and the evident mingling at this depth of different waters. In this connection the observations of Reese, previously referred to, in which it is shown that calcium phosphate in solution in carbonated waters is precipitated when the water stands over limestone, are particularly suggestive. As shown in my earlier papers the hard-rock phosphates of Florida are invariably formed directly upon limestone, and need not be sought for elsewhere. Moreover, they are thrown out of solution from carbonated waters which pass over and through these limestones, the manner of their formation apparently being entirely in accord with Reese's experiments.

The land-pebble phosphate deposits of Florida are probably derived, like the hard-rock deposits, from the Alum Bluff formation. The processes by which they have accumulated in their present form are, however, strikingly different. The hard-rock phosphates, as has been stated, represent chemical precipitates or replacement deposits, the phosphate having been transported to its present location in solution. The land-pebble deposits, on the other hand, appear to represent materials which are residual from erosion of the parent formation. The hard-rock phosphates occur in sections where the parent formation has entirely disintegrated over limestones. The land-pebble deposits on the contrary are found as a blanket deposit resting upon, and representing a concentration from, the parent formation. The matrix of the land-pebble deposits is, therefore, not strikingly different from that of the hard-rock deposits except as chemical action has modified the residue, particularly by the formation in many instances of siliceous boulders in the hard-rock deposits.

The grade of rock produced from the land-pebble deposits under the present methods of mining varies from 66 to 74 per cent. tricalcium phosphate, while individual samples contain 77 or 78 per cent.

Tennessee Phosphate Deposits

As further evidence of the complexity of the origin of phosphates in workable deposits, and of the diversity of ways in which they accumulate, may be mentioned the phosphates of Tennessee. Several distinct types of phosphate are found in Tennessee, only two of which, however, are being mined at present, namely, the blue rock and the brown rock. The blue phosphate occurs as a bedded deposit within, but forming the lowest member of the Chattanooga formation of Devonian age, which rests directly upon limestones of Ordovician age. The beds vary in thickness from zero to 50 in., although the bed of high-grade rock rarely exceeds 20 in. in thickness. In grade the blue rock varies from 30 to 85 per cent. The iron and aluminum in the better grade aggregate less than 3 per cent. The brown-rock phosphates, which at present are being more extensively mined than the blue, occur in irregular deposits lying near the surface and resting upon limestones. The brown rock is largely a shelly rock which breaks up into pieces of varying size. In addition, however, there is a considerable proportion of small pebbles which in mass have a dark brown color. The underlying limestone has an irregular top surface, and not infrequently projects into the phosphate stratum. The limestone is dense and has a bluish cast. It is more or less phosphatic. The age of this underlying limestone is Ordovician. The overburden consists of clay, some phosphate, and a limited amount of sand. It is reddish in color, and in thickness is entirely variable, averaging perhaps 8 or 10 ft.

The brown phosphates of Tennessee are very evidently formed *in situ* from phosphatic limestone. Hayes and Ulrich[9] find that at least four limestone horizons have given rise to brown phosphates in Tennessee. The calcium carbonate from the limestone is more or less completely leached out, and is replaced in part at least by calcium phosphate.[10] The rock is thus enriched and becomes a workable phosphate. The leaching of the rock usually begins along jointing planes and unchanged masses of the original limestone in this type of deposit frequently remain as "horses." Two types of deposits are recognized, which are described as blanket and collar deposits. The blanket deposits are those which extend over a considerable area. The collar deposits are formed where the phosphatic limestone comes to the surface around the slope of a hill. The collar deposits are necessarily limited in extent, while the blanket deposits may cover considerable areas. The brown phosphates, from their manner of origin, have necessarily accumulated in comparatively recent times.

[9] *Columbia Folio, U. S. Geological Survey*, p. 5 (1903).

[10] Some of the brown phosphate of Tennessee is formed, according to Dr. A. F. Rogers, by replacement of crinoidal limestone by calcium phosphate. (Personal letter, April 1, 1914.)

The blue phosphates, on the other hand, are much older than the brown, having accumulated in their present form during Devonian time. It is believed by Hayes and Ulrich that the blue phosphates were originally formed as residual material from, and resting upon, the Ordovician phosphatic limestones, much as the brown phosphates of the present time are formed. After this residual material accumulated the area was depressed, allowing the sea to cover the limestone. By the action of the waves in shallow water the residual mass was pretty thoroughly washed, the soil and clay material being sifted out and carried away while the phosphatic material was left to form the phosphate rock as found at present. The sea subsequently deepened, so that shales and other formations were deposited upon the phosphate.

The richest of the blue-rock deposits is believed to have been formed from the Ordovician limestone, known as the Leipers formation. This formation is full of the same minute spiral and other shells that occur abundantly in the immediately overlying phosphate rock. Phosphates were formed from Ordovician limestone other than the Leipers formation, but they are of a lower grade and at present are not workable.

It would thus seem that the blue rock of Tennessee was formed during the interval between the Ordovician and the Devonian, washed during the Devonian, and in this condition was preserved for modern mining operations.

Western Phosphate Deposits

The extensive phosphate deposits of the western United States are interbedded with sedimentary formations, and to this extent resemble the blue-rock phosphates of Tennessee. The rock in these deposits is described as prevailingly oolitic, although an exceptional occurrence is recorded by Richard and Mansfield[11] in which high-grade rock was found to consist of shell fragments regarded by Girty as broken shells of pelecypods. The source of the phosphoric acid and the history of its accumulation in the form in which it is now found in these deposits, if at present obscure, will perhaps, upon further investigation, become apparent.

Phosphate Deposits from Guano

The phosphate deposits of Navassa, a small island in the West Indies, may be mentioned as an illustration of those which are believed to have been formed from guano. In the case of phosphate deposits formed from guano, the phosphate is taken in solution by rain water, and after being carried to a lower level is redeposited, replacing the carbonate of the limestone. The rapidity with which this process may be carried on is illustrated by an instance cited by Dr. Albert R. Ledoux[12] in which

[11] *Bulletin* 470, *U. S. Geological Survey*, p. 376 (1911).

[12] *Transactions of the New York Academy of Sciences*, vol. ix, p. 85 (1890).

limestone on one of the south Pacific islands was observed to have been changed to phosphate to a depth of several feet within a period of 20 years, the phosphoric acid in this instance being leached by rain water from recently deposited guano.

Mining

Phosphate rock is mined either by open-pit or by underground mining. Those deposits having a removable overburden are mined by the open-pit method, underground mining being resorted to only for deposits interstratified with other formations, so that the overburden cannot be removed.

Underground Mining

The deposits mined in America by underground mining include the blue rock of Tennessee, the Arkansas deposits, and for the most part the extensive deposits of the western United States, which are as yet but little developed. In underground mining, ordinarily, operations begin at the surface outcrop of the phosphate stratum, the first rock being uncovered by stripping off the overburden. When the overburden can no longer be removed economically, drifts are run into the bank and the phosphate rock removed, support being given to the roof, when necessary, after the phosphate is taken out. This method of mining is similar to that used in mining coal seams. In the Arkansas and Tennessee mines the phosphate rock is first drilled and blasted. It is then broken up by pick and loaded nto tram cars to be drawn from the mine.

Open-Pit Mining

By far the greater part of the phosphate rock produced in America is obtained at present by open-pit mining, in which the overburden is first removed from the rock. The purity of the rock, however, is not materially affected by the methods employed in removing the overburden, and hence it is not necessary to describe these methods in detail. It may be noted, however, that diverse methods prevail, depending upon the thickness and character of the overburden, the magnitude of the operations and the facilities available. Examples may still be found of removal of overburden by pick and shovel, team and scraper, or team and dump cart. It is, however, only a shallow overburden that can be so removed profitably. In the larger mining operations the overburden is removed by steam shovel, by means of which the material is loaded into cars, which are then drawn to the overburden dump; or the overburden is removed by the hydraulic method, the material being pumped through pipe lines to the overburden dump. Whatever method is employed a limited amount of overburden remains with the phosphate rock which must be separated in subsequent treatment.

It is not necessary in this connection to describe in detail the methods of removing phosphate rock from the pit, since the purity of the rock is but incidentally affected thereby. It may be said, however, that the phosphate rock is either loaded by pick and shovel on wagons or tram cars to be drawn from the pit, or it is taken up by dredge or by hydraulicking (Figs. 1 and 2). If taken up by the hydraulic method the rock is forced through pipe lines to the washer plant. The character of the deposit determines the methods of removal that may be employed. Where the phosphate is loaded by pick and shovel, such objectionable impurities as siliceous boulders, limestone rock and clay balls are rejected at the pit, and in some mines only the coarse rock is taken, leaving the finer phosphate, clay and sand. As a rule, however, there is no attempt to separate the phosphate from the matrix before removal from the pit.

Preparation of Phosphate for Market

The phosphate rock mined by the open-pit method must be washed and dried. The methods of treatment described in this paper are those followed in America and particularly in the Florida mines.

Washing

The hard-rock phosphate of Florida when brought from the pit is dumped on to grizzlies with 2 or $2\frac{1}{2}$ in. openings. The fine materials of the matrix pass through while the coarse materials, including phosphate, flint, and limestone boulders, and clay balls, are lodged on the grating. The phosphate boulders are then thrown by hand into a rock crusher near by, while the flint and limestone boulders and clay balls are discarded. That part of the matrix which passes the grizzly together with the rock from the crusher is dropped into a log washer beneath. The practice in the land-pebble mines is somewhat different from that followed in the hard-rock section, the matrix, as pumped from the pit, being thrown as a rule into a large revolving tube, known as a separator, punched "hit and miss" with holes 1 or 2 in. in diameter. As the separator revolves the phosphate pebbles, as well as the finer materials of the matrix, fall through the openings and lodge on a screen beneath, while the coarser materials, including sand, rock, and clay balls, remain in the separator from which they are carried to the waste dump. From the screen beneath the separator the phosphate rock passes into the log washer. While this is the usual arrangement in the land-pebble phosphate mines, yet in some of the newer plants it has been found practicable to omit the separator altogether, the rock from the dump being allowed to enter the log washer after passing over a screen of about $\frac{1}{16}$-in. mesh. When the separator is omitted practically all the matrix from the pit passes through the log washers, and it has usually been found necessary in these plants to install a crusher, which is then placed between the

two logs. The larger pieces of bone and phosphate rock, as well as the clay balls, if not disintegrated by the washer, are broken up in the crusher, and the phosphate which they contain is saved (Fig. 3).

The log washer, through which the phosphate rock is passed, consists of two cylinders or logs placed side by side in a box or trough. A series of blades arranged in a spiral is fastened to each cylinder. The trough is inclined, the phosphate being run in at the lower end, and as the logs are made to revolve in opposite directions, the phosphate rock is pushed forward by the blades, meeting as it goes a constant stream of water. By this means the rock is pretty thoroughly washed, the water carrying all the finer materials of the matrix escaping at the lower end or in the newer washers through an opening at the side of the trough. Frequently the

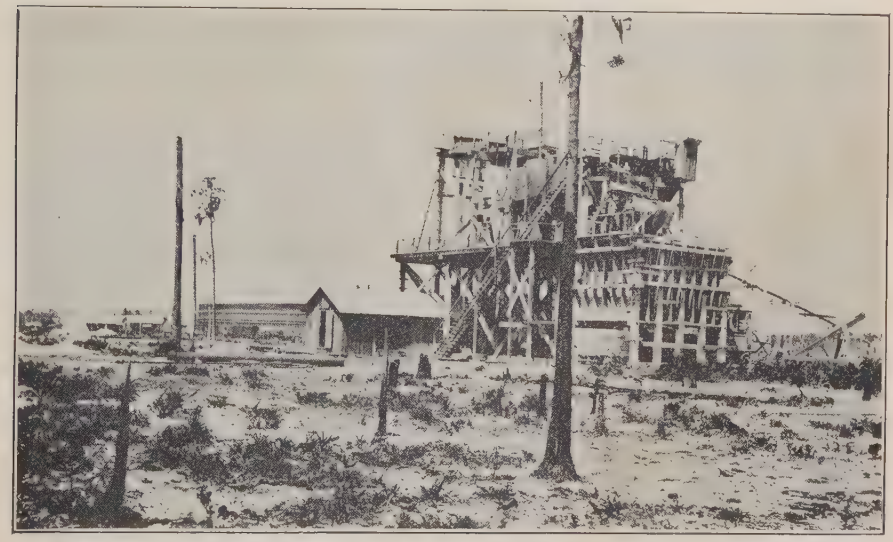

FIG. 3.—WASHER USED IN REMOVING IMPURITIES FROM PHOSPHATE IN FLORIDA.

phosphate rock is passed through a second log of the same type as the first, and in all cases receives a final rinsing while passing over screens.

In the hard-rock phosphate mines of Florida the coarse phosphate after leaving the rinser is made to pass over a picker belt, which is usually made in the form of a large revolving table. The phosphate rock remains on the picker belt during one complete revolution of the table, being carefully inspected by men and boys stationed around the table. The inferior rock, clay balls, flint and limestone fragments, so far as recognized, are picked out and discarded at this time, thus bringing up the grade of the shipment. In the land-pebble mines the phosphate rock from the last washer falls on jig screens, the finer of which are $\frac{3}{64}$ or $\frac{1}{32}$ in. mesh. From these screens the rock is elevated by endless cup chains to the loading bin.

Drying

After being taken from the pit the phosphate rock must be dried before being marketed. Two methods of drying are in use. The first of these, which is adapted to drying coarse rock, consists in piling the phosphate rock on ricks of wood. The wood is then burned, thus drying the rock. This method assists somewhat in cleaning, since clay and sand adhering to the rock tend, after drying, to loosen and fall away in subsequent handling.

The second method of drying, which is now largely used, is by the use of heated rotary cylinders through which the rock is passed. The rock is introduced usually at the cool end of the cylinder, and by means of various devices is made to pass through, escaping at the furnace or heated end. This method is adapted to drying small pebble rock; the coarser rock must be crushed before being dried by this method.

Improvements in Mining Methods

An important factor in the cost of producing phosphate rock is the necessity of discarding the low-grade rock, as well as that which cannot be properly cleaned or separated from the minerals with which it is associated. It is encouraging, however, to find that through improved methods of mining the amount of phosphate thus discarded is being considerably reduced, while the grade of rock produced is being maintained or advanced. In the modern phosphate-mining plants of Florida, practically no phosphate rock reaches the dump except that which will pass a $\frac{1}{16}$, $\frac{3}{4}$, or $\frac{1}{3\frac{1}{2}}$ in. mesh screen, or is carried out with the overflow from the side opening in the log washer. In Tennessee, where a very considerable part of the brown-rock phosphate is in the form of minute pebbles, settling tanks for saving the small pebbles have been introduced in late years. From the log washer the fine material passes through these tanks, and much phosphate that was originally thrown into the dump is thus cleaned and saved, and in fact some of the abandoned dumps in Tennessee are now being rewashed by this method.

However, notwithstanding these improvements in mining methods a very considerable amount of phosphoric acid in the form of very fine pebble and soft phosphate still finds its way into the waste dump. It is true that a large part of the phosphate that is thus lost is too high in iron and aluminum to be used in the manufacture of fertilizers under existing processes. It is, however, none the less desirable that the phosphoric acid be saved. To this end it is to be hoped that devices for recovering the very fine pebble phosphate may be further perfected, and that a new process of manufacture may be developed in which the grade of rock that can be used is not so strictly limited.

Tennessee Phosphate Practice

BY JAMES A. BARR, MT. PLEASANT, TENN.*

(Pittsburgh Meeting, October, 1914)

Geology and Mineralogy[1]

TENNESSEE phosphates are commercially divided into three varieties: Brown, blue, and white. The first two only are now of commercial importance. The white phosphates of Perry county are not being mined at present owing to their pockety occurrence. The white phosphate is

FIG. 1.—DEPOSITS OF 75 PER CENT. BROWN PLATE ROCK UNDERLAIN BY SAND.

the highest grade of the three varieties, running often more than 90 per cent. tricalcium phosphate.

The brown rock is found in three varieties, grading from the dense almost vitreous gray rock, 76 to 82 per cent. tricalcium phosphate, down through the ordinary cellular brownish gray rock, looking somewhat like

* Illustrations from photographs by J. S. Hook, Ithaca, N. Y.

[1] For a more complete discussion of the geology of Tennessee phosphates, refer to *Resources of Tennessee*, vol. 4, No. 2, article by J. S. Hook.

a sandstone, 72 to 76 per cent., to the lowest grade of soft mucky blackish brown rock, 72 per cent. and lower. The solid rock occurs in layers much broken up by joints filled with a corresponding grade of phosphatic sand, clay being the chief impurity.

The blue phosphate looks like a granular bluish limestone. It also varies as to color, the grayish variety being highest in grade, averaging 72 per cent.; the darker blue runs 72 per cent. and lower.

The brown phosphates of the Mt. Pleasant district occur in Bigby limestone of the Ordovician period. In Giles and Hickman counties the brown varieties occur in the Leipers limestone of the same period. Both limestones are a bluish crystalline variety often showing a banded structure upon weathering. These limestones are phosphate bearing often to the extent of 35 to 40 per cent., and are occasionally used as a low-grade fertilizer or for a filler for the manufactured fertilizer.

The principal deposits of blue rock, which are of commercial importance, occur between the Chattanooga shale and the Ordovician limestone, also phosphatic and of the same character as those just described.

Prospecting and Valuation

Brown Rock.—The extent, depth, and quality of the phosphate deposits are commonly determined by boring with earth augers and post-hole diggers 4 in. in diameter. Test pits are also employed in conjunction to give a better idea of the character of the strata.

In blanket deposits the holes are sunk 200 ft. apart in squares, while in hat-brim formations (ring deposits on a hill side) the holes are spaced 200 ft. on the long axis of the deposit and 50 to 200 ft. on the short axis, the latter spacing depending upon the width between upper and lower boundaries.

Since the overburden is not excessive, rarely being over 30 ft., the light and cheap 4-in. earth auger gives good results with two operators; $\frac{3}{4}$-in. pipe in convenient joints is used for rods. The post-hole digger is used for penetrating hard substances like flint or plate phosphate rock. It is merely a form of hollow drill, with a 4 by 12 in. barrel of $\frac{1}{8}$-in. steel. When penetrating phosphate a sample or disintegrated core is brought up by the tools and must be removed each time when full, necessitating frequent removals of the diggers from the hole.

About every 400 ft. a pit is sunk to obtain a sample of rock for recovery tests. This will be representative of the rock as mined by hand and sent to the washers, and will also give the true ratio of sand to lump rock with their respective analyses, which is of considerable importance.

The samples from prospecting are prepared for analysis by hand washing in an ordinary wash tub, saving the sand by decantation and separating the lump with the aid of a hose and a $\frac{1}{8}$-in. slotted screen.

The recovered products are dried, weighed, and analyzed separately. The percentage of recovery is calculated from weights in the usual manner. The weight of a cubic foot of phosphate in the ground should be determined so that the percentage of recovery as found may be converted into such convenient terms as 1 acre 1 ft. deep (43,560 cu. ft.).

Cost.—The cost of prospecting will average 7c. to 7½c. per foot of hole with augers or $1.50 to $2 per acre for average conditions.

Tonnages are usually calculated on the basis of a recovery of 1,000 tons[2] dry rock per acre-foot for high-grade deposits, but for the average

A. With 4-in. Post-Hole Digger.

B. With 4-in. Earth Auger.

Fig. 2.—Prospecting for Brown Rock.

grades this will be more nearly 850 tons per acre-foot. For more accurate work the percentages of recovery should be used as determined in the laboratory, allowing 15 to 20 per cent. for losses in actual mining and treatment.

Especial care is necessary in determining the average thickness of rock for unit areas when the deposits are known to lie in cutters. The phosphate will constitute from 30 to 50 per cent. of the total volume below the surface of the lime table.

The percentage of iron and aluminum oxides remaining in the washed product has an important bearing on the valuation of the rock. The usual

[2] A ton of dry phosphate is 2,240 lb.

guarantees are given below. For each per cent. in excess of guarantee, 2 per cent. of *BPL* is deducted.[3]

BPL Per Cent.	Sales Price f.o.b. Mt. Pleasant	$Fe_2O_3 + Al_2O_3$ Per Cent.
70	$3.00	6.5
72	3.50	5.5
75	4.00	5.0
76	4.50	4.5
78 to 80	5 00	4.0

The total final cost of dry rock f.o.b. cars may be estimated as $2.50, but with a good deposit and modern plant a cost of $2 should be obtained. The resulting profit as determined from the valuation of the rock less working costs should not be less than 20 per cent. per annum on the total investment, which will be: Property; this cost will average 25c. per ton of rock in the ground, or if on a royalty basis, 50c. Plant; capacity 75,000 tons per year, $150,000. Mining equipment, $30,000.

Blue Rock.—Since blue rock in the known commercial deposits lies mostly under a stratum of shale, the prospecting is necessarily done by diamond drills or drills of the Calyx type. Both give satisfactory results. The outcrops are usually exposed by shallow test pits. Since these deposits are continuous only in one direction, to any great extent, the long and short axes of the deposits are determined at the outset, by tracing the outcrops in gullies. This information materially aids in keeping down the cost of prospecting.

The minimum thickness of the vein for profitable mining at present prices is 24 in.

The sales price of blue rock f.o.b. cars Mt. Pleasant, Tenn., is: 68 to 72 per cent. *BPL* with 2 to $2\frac{1}{2}$ per cent. iron and alumina, $3.50 to $3.75. The lower grades of 62 to 68 per cent. *BPL* are usually sold ground 80 to 90 per cent. through 100 mesh at $3.50 to $4, of which 50c. represents the charge for grinding.

The cost of equipment and dead development work will be practically the same as given under cost of plants for brown rock, for a plant having a capacity of 300 tons in 12 hr.

The cost of blue rock f.o.b. cars with a modern plant averages $1.80 to $2 per ton, to which must be added with some deposits now being worked 25c. a ton to bring the price as quoted f.o.b. cars at Mt. Pleasant, because of extra freight rates.

Brown Rock Mining and Treatment

Removal of Overburden.—Occasionally narrow benches are stripped by hand methods. The dirt is more or less undermined by the removal of

[3] *BPL* is a local term for bone phosphate of lime or tri-calcium phosphate, $Ca_3P_2O_8$.

the underlying rock, and the bank caved over into the previously mined out bench by prying with a long rod from above.

Scattered deposits are stripped by scraper outfits such as are used in railroad work. The outfit usually consists of 5-wheeled scrapers with a hook team extra and a plough team. The work is contracted at 14½c. per cubic yard.

The major portion of the stripping is now done by Class 14 Bucyrus drag-line excavators mounting a 70-ft. boom with a 1½-yd. bucket. This machine is adapted to the removal of overburden that does not average more than 15 ft. With more than this depth the pits become too narrow at the bottom.

In Hickman county where the overburden averages 30 ft. a Class 24 drag-line machine is being installed. This has a 100-ft. boom and a 3½-yd. bucket.

The cost of operating the Class 14 machines is:

	Per Shift
1 Runner, $150 per month	$6.00
1 Fireman	2.50
1 Teamster	1.75
1 Ground man	1.50
1 Foreman	3.00
1 Team (owned by company)	3.00
Coal, 2 tons @ $2.40	4.80
Cable wear	5.00
Repairs, oil, and supplies	3.00
6 per cent. interest on $13,000. 250 shifts	3.20
10 per cent. depreciation	5.20
Total	$38.95

The above is for a machine having caterpillar traction for moving. If a timber and roller machine is used an extra ground man is required.

The capacity averages 1,000 cu. yd. per 10-hr. shift, although often 1,300 to 1,400 yd. are dug under favorable conditions.

While the larger-size machines average more in yardage the operating costs also increase so that the cost per yard is nearly the same.

Two mines use hydraulic stripping because the depth of the overburden is excessive in one and the mining conditions require it in the other. The cost averages 7c. per cubic yard.

The bank is cut down by a hydraulic monitor using a 2 or 2½ in. tip. The water pressure usually needed is 150 to 175 lb. per square inch. The water flows back from the face carrying from 10 to 20 per cent. solids, into a pump well. From the sump the water and overburden are pumped to the waste ponds by an 8-in. direct-connected motor-driven centrifugal pump. The pump requires from 1,500 to 2,000 gal. of water per minute for full capacity, with a motor power of 75 h.p.

Mining.—In the Mt. Pleasant field, the rock is mined by hand since the rock occurs partly below the top of the lime stratum, in cutters of irregular shape. Steam-shovel mining has been tried but is not favored because it does not permit hand sorting of clay and flint in the mines.

The rock, being soft and well jointed, is broken with an ordinary hand pick, and shoveled into 3-yd., 36-in. gauge tram cars. The miners use mostly a 10-tined fork where the cast is short but prefer a long-handled shovel when throwing up from the bottom of cutters.

The work is done chiefly by contract, the price being 25c. per tram when one handling only is required. Where two casts are required, 35 to 40c. per tram is paid. The miners keep the track up to the mining face.

Fig. 3.—Stripping with Steam Caterpillar Drag Line.

In Giles county some deposits are 20 to 30 ft. in depth without interfering lime boulders or cutters. These have been mined by steam shovel but results were not satisfactory because of impossibility of sorting out clay and flint at the mines, which resulted in low grade of rock at the washer. The grade of deposits, now being worked on the lower outcrop, should improve further back in the hill as is the case with most Tennessee rock. Hydraulic-mining methods seem more suitable than steam-shovel operations, in this case.

In Hickman county, because of the depth of the cutters, from 20 to 40 ft., hydraulic mining is successfully employed.

The same equipment is employed as described under hydraulic stripping.

FIG. 4A.—MINING PHOSPHATE IN CUTTERS.

FIG. 4B.—FACE MINING.

FIG. 4C.—MINING DEEP CUTTERS WITH CANTILEVER.

The centrifugal pumps used are of very rugged construction, being the same as employed in the Florida pebble district. The shell is of

FIG. 5A.—MINING DEEP CUTTERS, HICKMAN COUNTY.

heavy cast iron, and when worn out it is rejected, this being found more practical than the use of liners. The other wearing parts are the end plates, which are easily replaced. The shaft is protected from grit

FIG. 5B.—RECLAIMING PHOSPHATE SAND TAILING WITH 4-IN. STEAM-DRIVEN CENTRIFUGAL PUMPING OUTFIT.

getting into the stuffing box by a stream of pressure water. The stuffing box is submerged in a water seal to prevent air leakage. A bearing of

ample proportions, with ring oilers and oil reservoir, takes the weight of the shaft and overhanging impeller. The motor is connected to the pump shaft through a flexible coupling, the entire unit being mounted on one base plate. The motor is usually a three-phase, 60-cycle, 2,200-volt, wound-rotor type, operating at 500 rev. per minute with drum controller and grid resistances for continuous duty from half to full speed.

The pump is connected to the suction and discharge lines through heavy rubber tubing called flexibles. Steel tubing with peaned flanges is commonly used for all lines.

Transportation.—In the case of hand mining the loaded trams are hauled to the plant by saddle-tank locomotives, usually of the 17-ton class for main-line hauls and 12-ton for yard and mine switching. The

FIG. 6.—PHOSPHATE WASHER, SHOWING SETTLING TANKS FOR FINE SAND ON THE RIGHT.

17-ton dinkey can haul 30 cars on the average mine track while the 12-ton can handle 20 cars.

On both main line and mine sidings 40-lb. rails are mainly used, but 30-lb. rails are more often used in the mines.

Washing.—The tram cars are hauled up an incline by a belt-driven hoist to a tipple where the muck is discharged, with the aid of water, if necessary, into a crusher hopper.

The crusher is of the single-roll type, being more of a disintegrator and feeder. Enough water is sprayed in at this point to enable the crusher to pass a tram load through every 2 min.

The crusher discharges into the low end of a log washer, 25 ft. long, set on a slope of 1 in. per foot. The solids are carried forward and up the

slope by the paddles of the log, which are set on an angle of 45°, at the same time being subjected to a very thorough mixing so as to emulsify the clay, some of which is carried back with the water and phosphate sand to the tail of the log where it overflows an adjustable tail gate, into a launder leading to the sand washer.

Log No. 1 discharges into a second log washer, where fresh water is added.

Log No. 2 discharges into a conical trommel, 30 and 60 in. by 17 ft., having a $\frac{1}{2}$-in. metal jacket with $1\frac{1}{2}$-in. round perforations, which merely acts as a shell for the screen, and support for the inner lining jacket of $\frac{1}{8}$-in. metal having $\frac{1}{8}$ by $\frac{3}{4}$ in. slots. A 4-in. pipe extends through the center supplying water under pressure through $\frac{1}{4}$-in. holes; the water jets are directed against the rock, and effectually remove the undersize and remaining clay.

The undersize of the trommel goes to the sand washer. The oversize is discharged on to a picking belt where the clay, flint, and limestone are removed by boys. The picking belt discharges to the wet storage system.

The principal tonnage of phosphate rock is washed according to the above methods. One company uses drag conveyors with perforated bottoms, instead of logs, water being sprayed from above. Another company uses a revolving cylinder with flights serving to mix rock and water and elutriate the clay.

Sand Washing.—All the undersize from the log or rock washer department is treated in a separate sand washer. There are three distinct systems used:

1. The sand is washed in a drag classifier of the "El Oro" type locally called the riffle trough. It consists mainly of a double-strand chain flight conveyor with $1\frac{1}{4}$ by 30 in. flights traveling up a slightly inclined trough, about 40 ft. per minute. The sand pulp is fed near the middle and carried up past sprays of water. This system is not as efficient a washer as could be desired, has high repair costs and a low percentage of recovery.

2. Tank System. (*a*) The sand pulp from the rock washer is run into sand tanks about 10 ft. square and 30 ft. deep. When the tank is full the settled sand is washed out through a slatted opening into the sump of a steam pulsometer pump, which elevates the pulp to the next bin. This operation is repeated five times.

(*b*) Same as (*a*), but the operation is made continuous by hydraulic ejectors or jets, as they are locally called, to pump the sand, as it settles into the succeeding bin. This system avoids the use of expensive pulsometer pumps but requires more water, using 700 gal. per minute to pump 80 to 100 tons of sand in 11 hr.

3. The pulp is fed into the high end of an inclined revolving cylinder provided with internal flights for agitation of the charge. A counter cur-

rent of wash water is supplied to carry off the elutriated clay. The sand is discharged at the lower end by a spiral and further dewatered in an intermittently discharged settling box. The overflow from the settling

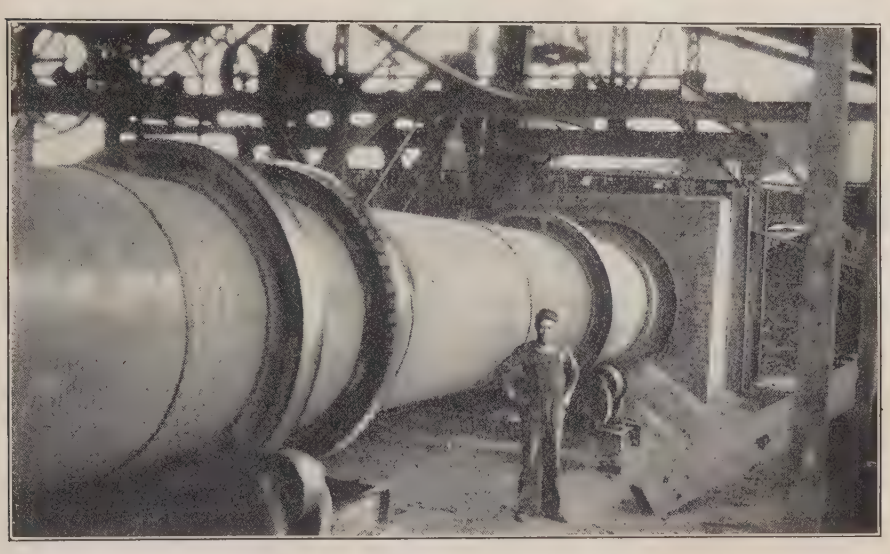

FIG. 7A.—MODERN ROTARY DRIER, 5 BY 40 FT.

box and cylinder washer is further settled in a tank with transverse baffles. The sand caught in this tank is washed out through holes in the side to a drainage pond.

FIG. 7B.—OLD METHOD OF DRYING LUMP ROCK ON PILES OF WOOD.

A new plant is now being designed that will use a dewatering tank, the underflow of which will be jetted into a series of Dorr classifiers, fresh water being added to the dewatered sand before feeding into the succeed-

ing classifier, the final dewatered product going to the wet storage system. Two of the rakes near the head end of each classifier will be replaced by spray pipes. This system has been tested out with excellent results and low water consumption. The overflow from all the above will be settled in a large 20 by 60 ft. tank. The product settled in these latter tanks will be washed separately either in Dorr classifiers or in Dorr thickeners with sprays on the arms.

Wet Storage and Drainage.—Since the material comes from the washers with 25 to 30 per cent. moisture, it is very important to provide a drainage system. By storing from two to three weeks the moisture content is reduced to 20 per cent., saving from 2c. to 2½c. per ton of rock in coal alone for drying, besides increasing the capacity of a 60 in. by 40 ft. dryer from 90 to 150 tons in 11 hr.

In the older style plants the storage is done in tanks or by building up in dumps with tram cars. The former is necessarily limited in capacity and the latter rendered expensive on account of the amount of labor used. In the modern plants, electric cranes operating clam-shell buckets are used.

Below is given a comparison of costs of the different types of cranes.

One 10-ton electric traveling crane 60-ft. span (with 2½-yd. clam-shell bucket)	$10,000
Steel runway 40 ft. to top rail, 400 ft. long	10,000
Generator capacity in power house, 40 kw. @ $55	2,200
Total	$22,200
One 30-ton, 8-wheeled locomotive crane, standard gauge, 40-ft. boom, 2½-yd. clam-shell bucket,	$14,000
Wooden trestle 12 by 400 ft.; material used 75 per cent. heart L. L. Y. pine, with concrete foundations	4,000
Generator capacity 40 kw. @ $55	2,200
Total	$20,200
One 30-ton steam crane same as above	$10,000
Trestle	4,000
Total	$14,000

With current at 1½c. per kw-hr. and coal at $2 to $2.50 per ton the steam crane can be operated as cheaply as the electric, where alternating current is used, but with direct current the cost is more in favor of the electric unit.

Drying System.—The next step after the storage of the material in the drainage piles is the reclamation of the drained phosphate and the drying of it in rotary driers or kilns.

The driers first used were light in construction, giving trouble from

break downs and rapid wear. They were from 52 to 56 in. in diameter and 32 ft. long. The drive was by friction only through the trunnions. Power consumed was high, averaging 30 h.p. per unit including fan. The material was fed into the hot end of the shell. The coal consumption averaged 90 to 100 lb. per ton of rock dried to 2 per cent. moisture, with a capacity of 80 tons per drier.

Modern driers, which are fed at the cold end, are characterized by heavy construction such as is used in rotary cement kilns. The size of the unit is usually 60 in. by 40 ft., and the power for driving 15 to 20 h.p., including fan for forced draft.

These driers reduce 135 to 165 tons of phosphate from 20 per cent. moisture down to 2 per cent., for the lower capacity given, and down to 3 per cent. for the higher capacity. The coal consumption averages 75 lb. West Kentucky coal per ton of rock dried.

The dried phosphate is elevated to a sizing screen and storage hopper, by an elevator, which is usually a single strand of No. 111 combination chain with buckets every fourth link. A traction head pulley and plain sprockets and idlers are used since the use of sprockets with teeth would cut down the life of the chain 75 per cent. Such an elevator lasts three years in ordinary service.

The phosphate is either chuted directly from the storage bin to railroad cars or else carried out to storage sheds by an electric-driven larry. The older plants use cable-driven cars or push cars.

By storing the rock, advantage can be taken of the residual heat to remove at least 1 per cent. of moisture. Cars are sampled at the works so that no credit is allowed for drying out in transit.

The rock is stored in a ground-level shed and loaded into cars by contract labor. This arrangement appears to be crude, but special conditions seem to warrant it.

Overhead reinforced concrete storage 10,000 tons capacity,	$80,000
	Per Ton
Cost of loading from above bin with car loaders	1.8c.
Interest 6 per cent. and amortization 6.7 per cent	10.2
Total	12.0c.

The above is based on a 15-year mine life with a capacity of 100,000 tons per annum.

Electric traveling crane	$10,000
Steel shed 10,000 tons capacity with crane way	25,000
Total	$35,000
	Per Ton
Cost of loading with crane and car loaders	3.9c.
Interest and amortization	4.4
Total	8.3c.

1 Steel shed, 10,000 tons capacity	$12,000
Or wood construction	$5,500

	Per Ton
Cost hand loading, contract price	7.5c.
Interest and amortization on $12,000	1.5
Total	9.0c.

The hand loading has the important advantage of being flexible and selective. The loader does not load wet streaks of rock; he throws out any foreign substance, spots and sweeps out his own car. He loads the car to the required capacity or is fined.

The automatic scraper and belt conveyor type of loading equipment would not be permissible on account of not being selective.

Production Costs.—The following were taken from the average costs of a mine during six months of good mining weather:

	Per Ton
Mining	$0.64
Transportation and team expense	0.23
Washing	0.46
Drying	0.47
Shipping and track expense	0.09
Total, per dry ton 2,240 lb.	$1.89

Mining and Treatment of Blue Rock

The outcrop is usually stripped and mined by hand, back to where the overburden does not exceed over 12 ft. on an average. For extensive outcrop mining a steam shovel is used for stripping.

The underground work is done on the room-and-pillar system, the pillars being recovered on the final round by drawing back. Main and cross entries are driven according to the topography, the rooms being led off from the cross entries alternately every 50 ft. The ordinary dimensions of the room are 25 by 200 ft. The long axis of the room is parallel with the rock jointing so as to obtain the maximum effect of the powder in blasting.

Advantage is taken of an overlying layer of low-grade kidney formation between the blue rock and the shale to make an overcut with air drills and to blast to this face. The skull is thrown back into the gob, acting as a support to the roof. Short timbers are used at the face to protect the workers from the shale roof, which tends to slab off.

The rock is loaded and mined by the contract system, into cars holding about 3,000 lb. The loaded trams are pushed to the room entry by hand where they are hauled to the mouth of the tunnel by mules. Here they are made up into trains and hauled to the plant by a 17-ton 36-in. gauge locomotive.

The surface plant is principally a crusher and drier. The blue rock is not subjected to any washing process.

The order of machinery, etc., in the plant is: Gyratory crusher; 60 in. by 60 ft. rotary drier fed at cold end; crushing rolls set to ½ in.; sizing screen; cable-driven storage larry.

The cost per ton for a modern plant producing 300 tons per 10 hr. is:

Working Costs

Mining	$1.30
Transportation	0.08
Crushing (gyratory)	0.03
Drying	0.08
Crushing (rolls), screening, and stocking	0.05
General works account	0.30
Repairs and depreciation	0.12
Total	$1.96
Loading into box cars	0.08
Freight to Mt. Pleasant	0.25
Total f.o.b. Mt. Pleasant (per ton, 2,240 lb.)	$2.29

Fine Dry Grinding, Both Blue and Brown Rock

A considerable tonnage is now being ground for use as a fertilizer direct, or for shipment to fertilizer factories for acidulation.

For phosphate containing lumps ½ in. or more in size primary coarse crushers are used. Following the coarse crushers a fine-grinding mill of the ring roll or Fuller-Lehigh type is used. One company uses a Hardinge mill. The discharge from the fine-grinding mill is elevated to sizing screens where the final size is taken out and the undersize returned to the mill.

Two companies now use tube mills after the fine-grinding mill as described above, to reduce the phosphate to the final size.

The raw rock, for direct application as a fertilizer, is ground to 90 or 100 per cent. through a 100 mesh; for acidulation, through a 60-mesh screen.

Manufacture of Acid Phosphate

Chemistry.—The final step in the treatment of phosphate rock for the fertilizer trade is its conversion from the insoluble tricalcium phosphate to the more soluble forms, by direct treatment with sulphuric acid.

The following are the basic reactions of the process:

(1) $Ca_3P_2O_8 + 2H_2SO_4 + 6H_2O = (CaH_4P_2O_8 + 2H_2O) + 2(CaSO_4.2H_2O)$.

(2) $Ca_3P_2O_8 + 3H_2SO_4 + 6H_2O = 2H_3PO_4 + 3(CaSO_4.2H_2O)$.

(3) $Ca_3P_2O_8 + H_2SO_4 + 6H_2O = (Ca_2H_2P_2O_8 + 4H_2O) + (CaSO_4.2H_2O)$.

Reactions (1) and (2) are the ones desired in the manufacturing process. With too little sulphuric acid present in the mix reaction (3) takes place, forming more or less dicalcium phosphate, insoluble in water. With an excess of acid, too much phosphoric acid is formed as in (2), resulting in a moist and lumpy finished product.

In determining the amount of acid required, account must be taken of the consumption of the principal impurities as noted below.

Carbonate of lime and magnesia are desirable to the extent of 5 per cent. because of the heat furnished in the reaction with the acid, the leavening effect of the CO_2 released and the drying effect on the final product in storage.

Fluorite is broken up by the acid, the resulting fluorine combining with the silica and passing off in part as the volatile H_2SiF_6. When troublesome, this must be properly handled and washed in scrubbers.

Silicates of iron and alumina form a gelatinous precipitate which coats some of the phosphate particles, protecting them from the action of the acid.

Iron and aluminum oxides are the chief constituents that must be carefully handled and regulated. They not only consume acid but result in a moist lumpy acid phosphate, if in excess of 4 to $4\frac{1}{2}$ per cent., unless special precautions are taken as to amount and strength of acid.

When the acid phosphate does not dry out well, its quality and per cent. of available P_2O_5 may be materially increased by adding raw, fine-ground phosphate rock before storage preferably, but also directly before mixing. Some advocate the use of ground limestone but this will cause some of the soluble phosphate compounds to revert to the insoluble, often to the extent of 2 to 3 per cent.

The manufacture of double superphosphates is done by filter pressing out the soluble portions of acid phosphate, and evaporating the filtrate in pans to 45 per cent. P_2O_5. The concentrated phosphoric acid is used to treat a fresh charge of raw phosphate rock:

$$Ca_3P_2O_8 + 4H_3PO_4 + 6H_2O = 3(CaH_4P_2O_8) + 2H_2O.$$

This is a highly concentrated product used mainly for export trade and the manufacture of baking powders, etc.

Manufacture.—The lump and sand phosphate is unloaded from the railroad cars by hand and wheeled in a barrow to the storage pile or to the coarse crusher. From the coarse crusher the phosphate is elevated into feed hoppers of the fine-grinding mills. The mills and the process used are the same as described previously under "Fine Grinding." A mill not previously mentioned, that is much used, with satisfaction, is the three-roll Griffin mill.

The ground phosphate is next treated in a mixing pan, in batches of 1 to 2 tons, the ratio of acid to rock being 1,000 lb. rock to 1,000 to 1,008

lb. 50° Bé. acid. One type of mixing pan is a round cast-iron pan revolving slowly, with one or two sets of stirring arms or plows, also revolving but at a higher rate.

When completely mixed, the contents of the pan are discharged through bottom doors into a square concrete or brick chamber below, called a den. Here the reactions continue aided by the heat.

After remaining in the den for about 12 to 48 hr., or until the mass becomes cool (150°F.), the contents are again discharged through bottom doors, by manual labor, into an elevating and conveying system that carries the acid phosphate to large storage piles.

The manufacturing process is carried on the year around, but shipment is made principally during three months in the spring and two months in the fall.

Before shipping to the consumer it is usual to add enough ammoniates and potassium salts to make up a complete fertilizer.

During shipping seasons the acid phosphate is reclaimed from the storage piles by hand labor and wheeled to the mixing and bagging system. This consists mainly of a disintegrator which breaks up the lumps of the acid phosphate; a screen takes the discharge from the disintegrator and returns the oversize, the undersize being sent to the mixing machine, where it is mixed with the proper amounts of ammoniates, potash salts and filler. The mixed product is bagged in a bagging machine, weighed and delivered to cars by hand.

It will be noted that there is a large amount of hand labor included in the scheme of operation that might be eliminated by properly designed machinery, and such is beginning to be included in modern plants.

The costs vary widely with the different freight rates on acid and rock. To the costs of raw material the additional charges for acidulation, mixing, and shipping are from $1 to $1.25 per ton. There is a loss of about 7 per cent. in the manufacture of the acid phosphate that must also be taken into account.

A plant with a capacity of 125 tons of acid phosphate per day will cost erected complete $100,000. However, the principal outlay in connection with the fertilizer business is the selling of goods on credit to farmers, which may amount in a business of usual volume to as much as $400,000, for a plant of the capacity mentioned above.

The present selling price of bagged acid phosphate f.o.b. cars Mt. Pleasant, Tenn., is $14 per ton for 14 per cent. P_2O_5 material, with $1 added for each unit above 14 per cent. The price of the complete fertilizer varies according to analysis, the base prices being $22.50 and $18. The European war is raising these prices.

Salt Making by Solar Evaporation*

BY W. C. PHALEN, WASHINGTON, D. C.

(Pittsburgh Meeting, October, 1914)

Salt-Making Processes

The production of salt in the United States divides itself at the outset into two distinct classes: (1) The mining of rock salt and its purification and separation into marketable sizes, and (2) the production of salt by evaporation from brines, bitterns, and other solutions.

The processes employed at the present time in the manufacture of salt by evaporation may be outlined as follows:

1. Solar evaporation.
2. Direct-heat evaporation, (*a*) in open kettles, (*b*) in open pans.
3. Steam evaporation, (*a*) in jacketed kettles, (*b*) in grainers.
4. Vacuum-pan evaporation.

In this paper only the solar-evaporation processes practiced in different parts of the United States will be described.

Solar Evaporation

Localities

In the Eastern States the solar-evaporation process is not generally employed for climatic reasons. In the Western States, particularly in California and Utah, the great bulk of the salt produced originates in the solar-evaporation process. Of the Eastern States producing salt, this process, so far as the writer is aware, is employed in New York State only.

New York[1]

Introduction

The manufacture of salt by solar evaporation began in the vicinity of Syracuse, Onondaga county, in 1789. A natural brine 68° to 80° salometer, with 17 to 20 per cent. sodium chloride, was and is still used.

*Published with the permission of the Director of the U. S. Geological Survey.

[1] In the preparation of this portion of the chapter on solar evaporation, the author has made free use of the works of F. E. Englehardt: *Bulletin No.* 11, *N. Y. State Museum*, pp. 38 to 44 (1893), and T. M. Chatard: Salt-making Processes in the United States, *7th Annual Report, U. S. Geological Survey*, pp. 506, 507, (1888).

The brine is stored in glacial gravels and was evidently formed by the circulation of ground waters through adjacent beds of rock salt.

But a small part of the evaporated salt manufactured in New York is now made by the solar process. Its manufacture is limited to Syracuse and vicinity, where it has survived from the early days of the industry. Though it has now lost some of its former importance, up to the year 1880, the time when the beds of rock salt in the western part of New York State began to be utilized, Onondaga county supplied the whole output of the State in rock salt. A large part of the total now returned for the county represents the salt in brine consumed in the manufacture of soda. The solar salt produced is mainly coarse salt and is used for practically the same purposes as rock salt. It is marketed through the Onondaga Coarse Salt Association.

The wells are located on lands once included within the Onondaga Indian reservation; and until recently the State supplied the brine to the individual plants, exacting a small tax on the product to cover costs of pumping and supervision. The lands and wells were sold in 1908 to private companies, namely, the Onondaga Pipe Line Co. and the Mutual Pipe Line Co. of Syracuse, the transfer being made for a nominal sum. This transfer definitely terminated the historic connection of the State with the salt business.

Solar salt is made from a natural brine—the only instance of its use in New York. The process, of course, is carried on only during the spring, summer, and fall months, usually from the middle of March to the middle of November, depending on the weather.

Method of Manufacture

Former Method.—The manufacture of solar salt in New York was formerly carried on in shallow wooden vats or "covers," provided with light movable roofs running on rollers. During the evaporation season the roofs were removed to one side, but replaced during rainy weather, and at the end of the salt-making season. There were ordinarily three sets of vats or "rooms," known respectively as "deep rooms," "lime rooms," and "salt rooms."

The deep rooms served for the reception of the brine from the well or pump house. When first pumped the brine was usually perfectly clear, but with the escape of gases (CO_2), ferrous carbonate separated, which oxidized to the hydrated ferric oxides, and settled out as a yellow mud, leaving the clear brine.

The lime rooms were constructed nearer the ground than the deep rooms, the clear brine being drawn from the higher deep rooms into the lower or lime rooms. Evaporation continued in the latter until the brine became saturated and salt crystals began to separate. During this

process, particularly when the brine was near the point of complete saturation, gypsum began to separate. The brine, now fully saturated and known as "pickle," went to the lowest set of rooms, known as the salt rooms. Here the salt and the remainder of the gypsum separated, sufficient pickle being added from the lime room from time to time to keep the salt crystals well covered. The term "lime room" in the above descriptions is somewhat misleading as no lime was used in the process. When sufficient salt collected in the salt rooms it was harvested. This took place generally three times during a season.

Present Method.—The process described above was the original method of solar evaporation. Great improvements have been made in it, one of the greatest being the replacement of the deep rooms and lime rooms by "aprons," whereby the evaporating surface and the yield of salt per cover (288 sq. ft.) have been greatly increased. The aprons are large troughs from 15 to 20 ft. wide, about 3 in. deep, and varying greatly in length. They are generally erected over the deep rooms, serving as roofs, thus making store rooms for the saturated brines of the deep rooms. They are built on piles or posts, and are so constructed that brine or rain water falling on them flows toward two sets of holes provided with wooden plugs, the grade of flowage being 1 in. per 100 ft. One set of holes communicates with the deep rooms. The brine run on to the aprons often becomes saturated in one good drying day. It is then discharged into the deep room below and its place taken by a fresh portion of brine. When rain is expected the plugs over the cisterns, or deep rooms, are drawn and the brine which flows into them is thus saved from dilution. The plugs are then replaced and the other set removed, thus allowing the rain water to flow to waste. When clear weather returns, the partly concentrated brine is returned to the aprons by pumps.

"The great advantage of this improvement lies in this—that the brine is concentrated and purified while being transported to the covers; the great length and width of the aprons, the shallowness of the layer of brine and its complete exposure to the sun, air, and wind greatly facilitate purification and evaporation while permitting the use of the entire number of covers for salt making, instead of having as formerly one-third of the total number taken up for lime rooms and deep rooms." In this way the production of solar salt during a given season has been greatly increased per cover. As the capital invested in covers is considerable, the economy of the process is readily seen. (Figs. 1 and 2.)

Utah

The Water of Great Salt Lake

Salt is obtained from the water of Great Salt Lake. According to different authorities who have analyzed the lake water, the chief differ-

FIG. 1.—Yard of Salina Salt Co., Syracuse, N. Y.

FIG. 2.—Loading Wagons, Salina Coarse Salt Co., Syracuse, N. Y.

FIG. 3.—Harvesting Salt, Inland Crystal Salt Co., Saltair, Utah.

ences in the results are in the degree, but not the character, of the salinity. It is to be expected that the former would change with greater or less dilution, as the waters of the lake rise or fall, from local or other causes. Although the degree of salinity of the lake is variable, ranging approximately from four to seven times that of ocean water, the composition of the saline matter contained in it is much like that found in ocean water. The following analyses show the composition of the salts contained in the water:

Analyses of Salts in Water from Great Salt Lake[2]

	1	2	3	4	5	6	7	8
Cl	55.99	56.21	55.57	56.54	55.69	55.25	55.11	53.72
Br	trace	.....	.....	.....	trace	trace	.....	.....
SO_4	6.57	6.89	6.86	5.97	6.52	6.73	6.66	5.95
CO_3	.....	0.07	.....	.....	.....	.....	.....	.....
Li	trace	.....	.....	.....	0.01	trace	.....	.....
Na	33.15	33.45	33.17	33.39	32.92	34.65	32.97	32.81
K	1.60	(?)	1.59	1.08	1.70	2.64	3.13	4.99
Ca	0.17	0.20	0.21	0.42	1.05	0.16	0.17	0.31
Mg	2.52	3.18	2.60	2.60	2.10	0.57	1.96	2.22
Fe_2O_3, Al_2O_3, SiO_2	.....	.....	.....	.....	0.01	.....	.....	.....
	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Salinity, per cent	14.994	13.790	15.671	19.558	[a]23.036	27.72	22.99	17.68

[a] More correctly, 230.355 g. per liter.

Compared with the analyses of sea water, the sodium is higher, and the lime and magnesia, especially the former, are distinctly lower. Carbonates are nearly entirely absent, but are present to an appreciable extent in ocean water. There are other variations in composition between lake and ocean water, but in general they are of the same type. Additional analyses appear in a monograph by Gilbert.[3] Gilbert states (p. 253) that the quantity of sodium chloride contained in the waters of the lake is about 400,000,000 tons and that the amount of sodium sulphate is about 30,000,000 tons. In the same volume Gilbert also considers the different surveys made of the lake, the depth of water, the oscillations of its surface, the changes and the causes of change in its area, the sources of its saline matter, together with the rate and period of its accumulation.

The Salt Industry

Salt has been made by the solar-evaporation process for many years along the shores of Great Salt Lake, the industry dating in fact from the early settlement of the region. A small amount of salt is reported as

[2] Taken from F. W. Clarke: Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 144.

[3] Gilbert, G. K.: *Lake Bonneville Monograph, U. S. Geological Survey*, vol. i, pp. 253 to 254 (1890).

having been made recently at Promontory Point, Box Elder county; Withee, Weber county; Syracuse, Davis county; and at Garfield, Salt Lake county. Fifteen miles west of Salt Lake City, near Saltair, is the plant of the Inland Crystal Salt Co. The operations at the latter place are by far the most elaborate ever attempted along the lake shore. The process of obtaining salt near Saltair is described below.

The brine is pumped from the lake into a flume in which it is carried about 3 miles to the ponds, where concentration and crystallization are effected. The ponds are designated from the processes taking place in them as: Settling ponds, stock or evaporating ponds, and crystallizing or harvesting ponds. The ponds are separated from each other by clay embankments held in place by boards set on edge. These clay levees are 2 ft. wide and 22 in. high. Pumping begins in April and continues until the first of September, excepting during times of storm. Evaporation goes on approximately at the same rate as pumping, which is about 5,000 gal. per minute, averaging 16 hr. per day from the middle of June to the middle of September—the season of greatest dryness and hence of maximum evaporation. Pumping is regulated so as to maintain the level of the brine in the ponds.

The brine from the lake goes first to the settling ponds, where it is allowed to remain five to six days. Here all suspended matter is deposited. The settling ponds are estimated to be about 75 acres in extent. From the settling ponds it goes to the stock ponds, which cover an area of about 1 sq. mile. Here it remains until concentration reaches complete saturation and salt is ready to deposit, the length of time depending upon the weather. The brine then goes to the harvesting ponds. During the summer of 1912, when the writer visited the region, 30 days elapsed between the beginning of pumping operations and the time the brine reached the harvesting ponds. The brine is conducted from one pond to the other by gravity, the flow being controlled by means of small gates.

In the harvesting or crystallizing ponds, as the name suggests, evaporation is carried to the point where all salt separates out. To insure a clean product, an underlayer, or salt floor, is allowed to crystallize out each year, unless it is left over from the operations of the preceding year. This underlayer serves as a clean floor upon which the subsequent crystallization takes place. When it remains from the previous year, as happens when the salt crop is larger than the market demands, the salt in it becomes more or less dirty. It is then used as stock salt though it is a first-class product in every respect. The salt which is marketed comes from the upper layer, the plane of demarcation between it and the floor being known as the "split." The average crop is about 3 in. of salt. The minimum crop is about 2 in.— the maximum usually 4 in., but a crop of nearly 5 in. has been obtained. Three inches is considered a good crop.

During the course of the salt-making season the bittern formed in the harvesting ponds is drawn off twice, once during the middle of the season, and again at its end. In this way the salt is freed from the bulk of magnesium salts and sodium sulphate. Care must be exercised to draw off the bittern toward the season's end, before cold weather begins, to insure against precipitation of sodium sulphate, which crystallizes out when the water reaches a temperature of 20° F.[4]

When the bittern is drawn off for the last time and the salt is ready to be lifted, it is loosened by means of ordinary plows drawn by horses. (Fig. 3.) It is then stacked by means of scrapers, wheelbarrows, or hand cars run on tracks into large piles alongside the railroad which traverses the salt field. The salt is conveyed from the stacks to the cars, which are transported in turn by means of an oil-burning locomotive to the company's mill, where the salt is further refined A portion of the yield may be sold without further treatment, for the different purposes for which rock salt is used.

California

Localities.—More than 97 per cent. of the output of salt in California originates from the evaporation of sea water along the coast. The main

Fig. 4.—View of Salt Ponds. Oliver Salt Co., Mount Eden, Cal.

production of this solar salt comes from along the east and west shores of San Francisco bay, in Alameda county, and San Mateo county, respectively. In Alameda county the centers of production are near Alvarado, Mt. Eden, Russell, and Newark; in San Mateo county, solar salt is produced a short distance south of San Mateo and near Redwood City. Other places where salt is produced by solar evaporation are near Long

[4] Gilbert, G. K: *Loc. cit.*, p. 253. Talmage believes the critical temperature of the separation to be within a few degrees of the freezing point of fresh water, say approximately at 35° F.

Beach, Los Angeles county, and on San Diego bay, San Diego c
In all these places the operations are conducted along similar line
they differ considerably from each other in the details worked o
individual operators in charge. There is also considerable diversity in nomenclature connected with the processes in their various stages. In the following account it is aimed to give general descriptions of processes with certain details observed at representative plants in the different salt-making localities.

Salt-Making Season.—The concentration of the water in San Francisco bay is variable, depending on the season of the year and other factors.

FIG. 5.—SALT CRYSTALLIZED IN A CRYSTALLIZING POND. LESLIE SALT CO., SAN MATEO, CAL.

The dilution of the bay water is caused in part by the fresh water from the San Joaquin and Sacramento rivers and is increased after winters of heavy snow which is not melted away completely until the summer is well advanced. Brine is taken into the salt ponds from about the middle of May to the middle of October, but the strongest brines are obtained after July first, as the dry season lengthens and the inflow of fresh water from the rivers and streams is appreciably reduced. Generally the salt-making season is over by Sept. 15, as the rainless season lasts usually from May to September. A season as long as 210 days without rain has been known, however, along San Francisco bay.

The season for 1911 may be taken as a typical one. During the summer of that year the evaporation and rainfall as observed at one plant were as follows:

Rainfall and Evaporation in Inches, Season, 1911

	Rainfall, Inches	Evaporation, Inches
April	1.36	$3\frac{3}{8}$
May	1.14	$5\frac{5}{16}$
June	0.67	$6\frac{10}{16}$
July	none	$7\frac{13}{16}$
August	none	$7\frac{13}{16}$
September	none	$4\frac{15}{16}$
October	0.77	$2\frac{15}{16}$

The water from the bay or ocean is not taken into the works continuously, but usually on two to six days each month, depending on the season when the tides are at their highest. Water is usually taken in at

FIG. 6.—GRAVITY METHOD OF RUNNING BRINE FROM ONE POND TO ANOTHER. OLIVER SALT CO., MOUNT EDEN, CAL.

the period of the new moon. Sometimes the high tides come twice a month, but once a month is the more usual. The periods of highest tides are ascertained from the tide book issued by the U. S. Coast and Geodetic Survey. During the periods of high tide, water is not running into a salt works continuously, but only from $1\frac{1}{2}$ to 3 hr. during a given high tide. These are statements based on general practice, but it is understood that at some plants, where a slough runs through the intake pond, salt water may be taken into a given system at any time or every day. Thus, at Long Beach the sea water runs into a ditch at every high

tide, that is, twice each day. Salt water may therefore be pumped continuously day or night from this ditch into the plant.

In southern California, naturally, the salt-making season is longer than it is near San Francisco. At San Diego it is estimated to be about seven months, namely, from May to November. On San Diego bay the salt water also may be taken in continuously during the salt-making season. This is accomplished by draining the tide ponds, whereby water may be taken in on every tide above a given height.

From 1908 to 1912, the mean average date when crystallization of salt began along San Francisco bay was as follows: 1908, May 1; 1909, May 13; 1910, May 3; 1911, May 25; 1912, April 18. The reason for the early date in 1912 was on account of the very few late rains during that year. Records are closed, that is, evaporation readings are stopped, each Nov. 1, when the days are so short and the weather so cold and uncertain that there is practically no evaporation.

Strength of Brine Used.—The average density of sea water is 1.027, corresponding to a salinity of 3.72 per cent. regarded as pure salt. The three samples of bay water collected as they ran into plants on the shores of San Francisco bay and on San Diego bay gave the results indicated in the following table:

Density and Salinity of Water used in Salt-Making Process on the California Coast

No.	Baumé Degrees	Specific Gravity[a]	Percentage of Salt[b]	
23	3.53	1.025	3.44	Water from San Francisco bay, Mt. Eden.
28	3.12	1.022	3.34	Water from San Francisco bay, Alvarado.
33	4.10	1.029	4.00	Water from San Diego bay.
..	....	1.027	3.72	Sea water.

[a] Specific gravities were taken at 22° C., but are referred to water at 15° C.
[b] Assumed to be pure NaCl, which is not the case.

Thus it will be observed that the normal salt water entering the salt plants along San Francisco bay is less concentrated than ordinary sea water, while the opposite is true of the bay water near San Diego. These samples of bay water were collected: No. 23, Oct. 5; No. 28, Oct. 8; No. 33, Oct. 17; *i.e.*, on days well advanced during the dry, or salt-making season. One of the reasons for the difference in density between the waters from San Francisco bay and San Diego bay is that farther south the warmer and drier season lasts much longer. The evaporation therefore is more intense, hence at a given date during the dry season the southern waters would naturally be expected to show a higher degree of salinity. Other conditions which tend to reduce the salinity of the San Francisco bay water are the streams of fresh water which flow into it.

Ponds.—The salt-making operations are conducted in what are known as ponds, designated from the operations conducted in them. Thus

there are: (1) The storage, the intake, receiving or tide ponds into which the salt water is received from the bay, (2) the concentrating ponds, and (3) the crystallizing ponds. By some operators all the ponds between the tide ponds and the crystallizing ponds are known as secondary ponds. Technically they may be known as pickling ponds. The term "lime ponds" is sometimes applied to those in which the bulk of the gypsum crystallizes out.

The salt or ocean water comes into the works generally through a slough, passing thence into the intake, receiving, storage, or tide pond, as the first pond is variously called. This pond is provided with large flood gates with automatic adjustments to open them to allow the water to run in, but which close when the tide begins to ebb. The salt water remains in the intake pond a variable length of time, in the meanwhile evaporating. It is pumped out before the next salt water is taken in. It then goes forward through the different ponds becoming more concentrated, until it finally reaches the crystallizing ponds. It is run into these latter when it reaches a strength of about 25° Bé., or when salt begins to form, usually to a depth of about 6 in. At some plants, when it reaches a strength of 29° Bé., the bittern with some salt still in it runs over into other ponds. Here it evaporates until a concentration of 32° Bé. is reached, when all the salt is out of it (see table below). The mother liquor remaining is then run into the bay and goes to waste at many plants, but it seems that some use should be made of the magnesia and potash salts contained in it. At some plants a part of it at least is saved and utilized as described later in this paper.

Specific Gravity of Bitterns from Crystallizing Ponds

No. of Sample	Baumé Reading at 22° C.	Specific Gravity[a]	Locality
25	32.50	1.286	Oliver plant, Mt. Eden, Cal.
26	29.60	1.256	Leslie Salt Co., San Mateo, Cal.
30	27.21	1.231	California Salt Co., Alvarado, Cal.
31	28.00	1.239	Long Beach Salt Co., Long Beach, Cal.
32	28.60	1.245	Long Beach Salt Co., Long Beach, Cal.
34	27.50	1.234	Western Salt Co., San Diego Bay.

[a] Specific gravities were taken at 22° C., but are referred to water at 15° C. They were made by R. K. Bailey, of the U. S. Geological Survey, with a Westphal balance.

Area is one of the dominant factors in the industry. There are plants where the total acreage of ponds is as low as 500 acres, but others are reported to have more than 2,000. The ratio of the acreage of crystallizing ponds to that of the whole system was found at two important plants to be as 1 to 10, but the older the plant, the more acreage it has for crys-

tallizing purposes, the reason being that the ground becomes thoroughly salted down with continued usage.

The ponds have mud floors. The salt in places is harvested to within 2 in. of the mud floors and removed to the main stock piles. The remaining 2 in. is shoveled up into smaller piles and this grade is used as ice cream and stock salt.

The ponds are separated from each other by low embankments known as dikes. The width and height of these dikes vary greatly. They may be as much as 3 ft. above the marsh level and 1 or 2 ft. above brine level Some are very narrow, others, especially some of the outside dikes, are wide enough and sufficiently well constructed to serve as roadways over which teams and autos may be driven. The low narrow dikes or levees are not considered by all the salt makers as good practice as they require constant attention and repair owing to wash by the waves. Where the

FIG. 7.—SLOUGH AND WIND MILLS. OLIVER SALT CO., MOUNT EDEN, CAL.

levees are too high the evaporating action of the wind is interfered with, but at San Diego there is not much wind. The dikes between the ponds were formerly flanked with sod, but this practice is now being done away with.

Methods of Pumping.—Pumping along San Francisco bay is done in part by Archimedes screws propelled by wind mills; also by means of paddle wheels. The average plant may have as many as 40 wind mills. Some plants use gasoline engines as an auxiliary power, especially in the fall when the trade winds die down. In August, the trade winds slacken or blow intermittently and therefore cannot be depended upon. The wind mills are set solid to the northwest, from which the trade winds come, and from their manner of construction they cannot turn around.

The wind mills are generally placed so as to pump from the outer

or intake pond into the next pond. From the latter it is sought to have the brine move along by gravity, the flow from one pond to the next being controlled by small gates. Wind mills or other means may, however, be used for lifting in other parts of the system, when gravity is found to be inexpedient.

Harvesting.—The crop of salt of the preceding season is usually sold by July 1 of the following season. Enough salt is then lifted from time to time to supply the current demand until Sept. 1, when harvesting is pushed as vigorously as possible. At many plants the first salt is lifted

FIG. 8.—METHOD OF LIFTING SALT IN HARVESTING POND. CALIFORNIA SALT CO., ALVARADO, CAL.

about June 1, and the operation continued until about the first of January or February of the following year if the crop is a big one and labor is rather scarce. Ordinarily, however, lifting is finished by December. The salt is lifted by hand into small tram cars each holding usually about 1 ton. The salt thus lifted is pinkish in hue and the mother liquors are likewise pinkish. A small locomotive propelled by gasoline or electric power picks up eight cars, more or less, and hauls them to the mill; or they may be hauled to the salt piles or stacks and dumped. There may be one huge stack near the mill itself from which salt is used according to the demand. The latter method perhaps may

be the more economical if practicable, for it reduces the surface exposed to solution from rains and also the surface which has to be broken up in the subsequent utilization of the salt. This latter is an important consideration, for the surface hardens so firmly that it is broken up only with the greatest difficulty; and in some cases the cost of breaking up a particularly old crust on a small stack costs more than the salt in the stack is worth. Sometimes old and small stacks may be discarded completely. In some places after standing in the stock piles the salt hardens so firmly that crosscut saws have to be used. These saws cut it into 4-ft. lengths which are pried apart and then crushed in the subsequent milling operations.

FIG. 9.—GENERAL VIEW OF CRYSTALLIZING POND AND THE LAYING OF TRACK. LESLIE SALT CO., SAN MATEO, CAL.

When the salt harvested is not taken directly to the mills, but to the stacks, it may be manipulated in various ways as outlined below. After reaching the works it may be dumped through a hopper on to a screw conveyor, passing thence to a bucket elevator, which lifts it to the main salt stack. Not every plant can use this method, because the ground has to be very firm to accommodate the enormous tonnage of one main stack, a condition not easily obtained in regions of marshy land. Sinking of the ground may occur and an appreciable part of the product be lost. An instance is known where 1 ft. of filling sank each year for a period of 20 years.

Instead of shoveling the salt into small cars to be hauled to the stacks, dumped and there elevated, a much simpler and comparatively inexpensive

process may be used, in which the salt is shoveled by laborers, a line of whom extends the width of a pond, on to an endless belt 20 in. wide, running on rollers, which conveys the salt to one side of the pond. The power is furnished by a gasoline engine, but an electric motor might serve as well, or better. As the salt leaves the belt it is washed by a spray of salt water from the bay. It is next dragged up a perforated copper screen, during the ascent of which it is again washed with bay water. The endless drag chain is provided with forks to break up the larger lumps. The salt is then elevated to the stacks by a pan elevator run so that the open mouths of the pans are reversed to enable the excess of wash water to drain. By a washing device the main belt which conveys the salt from the pond is kept perfectly clean.

FIG. 10.—VIEW OF SALT POND IN WHICH THE SALT IS BEING HARVESTED. LONG BEACH SALT CO., LONG BEACH, CAL.

The method just described has many labor-saving advantages. As soon as a given area of salt has been removed from the crystallizing ponds the entire layout is moved forward on tracks provided for the purpose, and the salt pile may thus stretch out the length of the pond. The washing of the salt likewise is performed at a time when the adhering bittern is most readily and completely removed. The salt from the outside stock piles is brought up to the mill as needed in 5-ton cars and there subjected to the various operations such as crushing, etc., to fit it for the finer uses, or it is re-dissolved and prepared for the vacuum pans.

At Long Beach the salt is shoveled from the stock pile into small cars which are trammed by a rope tram to a hopper through which it passes, being ground in the process. It is next elevated by a bucket conveyor and dumped into a bin, which in this case is an ordinary freight

car fitted up for the purpose. When the bin is filled the car is motored to the mill.

At San Diego the salt has to be picked as it lies in the crystallizing pond before it can be lifted with shovels. The greater firmness of the crystallized salt as compared with that along San Francisco bay, which makes picking necessary, is possibly due to the greater rapidity of crystallization. It is customary to begin harvesting just as soon as the bittern is run off the crystallizing ponds. This is due to the fact that the salt is easier to pick, shovel, and wash at this stage than at any other.

It is difficult to give the amount of salt which can be made during a given season, as the quantity is variable from year to year, depending upon the weather and other factors. Perhaps a thickness of 5 or 6 in. of salt per season would be a fair approximation for the region along San

FIG. 11.—SALT PILE AND PLANT. LESLIE SALT CO., SAN MATEO, CAL.

Francisco bay, but in the southern part of the State, owing to the longer and drier season, it would be very much larger. At San Diego, for example, an average harvest is 6 in. of salt, but two such harvests are possible per season, in this region. Hence it is possible to make 12 in. of salt per year in this locality.

Waste Bittern.—The bittern from the crystallizing ponds has never been saved at many of the plants in the past and at most of them it is even now running to waste. Since the importance of potash salts has come to be realized some of the operators have in mind the utilization of the potash content of the bitterns, but at most of them nothing definite with regard to such use has yet been evolved. At one plant a small part of the bittern is saved, refined, and used for medicinal purposes; at an-

other it has been used in the manufacture of "wood stone" and in the manufacture of magnesium oxychloride cement. The latter substance has been found very resistant to the passage of electricity, and hence has been applied in the manufacture of switch boards. It is understood that the bittern has been used by the Santa Fe Railroad to lay the dust along its road bed, but experience has proved it to be too costly for this purpose, as it has to be renewed too frequently.

Milling.—From the nature of the processes employed in the manufacture of solar salt, and more particularly from the differences between them and those employed in other parts of the United States, the first stages of the milling operations are different in California from the practice elsewhere. When the salt is lifted from the crystallizing pond, it is contaminated with considerable adhering pickle or bittern and dirt of various kinds. If it is not to be used at once it may undergo a preliminary washing, after which it is conveyed to the stacks as already indicated under harvesting. If it is to be used immediately it is transported to the mill and there washed free from adhering impurities. This is accomplished in various ways. It may be dumped into a hopper and a current of hot brine from the pickling pond poured over it. This brine from the pickling pond represents bay water concentrated to complete saturation and from which no salt has been crystallized. For a concentrated solution it contains the least amount of mother-liquor salts. Its use results in the solution of a minimum amount of salt during the washing process. From the hopper the salt is lifted in an elevator, draining in the meanwhile. It is then passed through rolls where it is crushed into "half-ground salt" or "three-quarter ground salt." It then goes forward to other vats, where there is an artificial brine made from salt and fresh water and in which consequently there are no mother-liquor salts. The salt on removal from these vats is stacked in heaps to drain. The coarse salt is then sacked for one branch of the trade. If it is to be subjected to further refining processes the salt goes from the vats to a centrifugal machine where adhering water is removed. It is then conveyed to driers where it is thoroughly dried by steam heat. The driers are elongated cylindrical affairs containing steam coils. They are provided with fans to pump warm air through them, which greatly assists in the drying operation. From the driers the salt goes directly to the rolls where it is crushed. From the rolls it goes to the sifters where it is graded according to fineness, depending on the purposes for which it is to be used. It is then ready to be sacked for the trade. At different plants the operations differ greatly in details to suit individual tastes and those of the customers.

At some of the plants the solar salt in the stock piles is re-dissolved and the finer grades are made by the vacuum-pan process. When the vacuum-pan process is employed the manipulation of the salt at the mill may be quite different from that outlined above.

Discussion

David T. Day, Washington, D. C.—Consider the United States broadly and the distribution of the salt industry. Think of the old solar-evaporation process up in New York State, then of the more recent processes in Michigan and Kansas and Louisiana, where the rock salt furnishes a large industry, then of the solar-evaporation process at Salt Lake, then turn to the crucial place in the United States where there is an embarrassment of salt, and that is California. I am not forgetting the immense body of salt east of that in the Virgin River country of Utah; but in California the situation is interesting. There are very large quantities of salt in the Searles Marsh region; again, in Southern California, at the southern end of Death Valley, there are two or three mountains of salt which have been explored recently, where the salt is sufficiently pure to be used at once. There is also a large deposit of gypsum, and it would be possible there, by extremely cheap means, to produce an amount of salt sufficient to furnish the United States' greatest needs. The principal reason that capital has been deterred from developing such a large mass is not the fact that solar evaporation produces a great deal of salt around San Francisco, but the fact that if potash salt is to be produced in connection with the salt in the borax region a great deal of ordinary salt must be disposed of. It cannot be dumped, for it would get back into the brine again; the first rains would bring that salt back into solution again. It must be sold, given away, or hauled to a distance, which would mean a considerable freight charge. The salt will be going begging.

E. G. Spilsbury, New York, N. Y.—The remark of Dr. Day's regarding the likelihood of the production of a large amount of waste salt brings to my mind the condition that obtains in Germany in all of the potash fields.

In Germany salt is a government monopoly. In the development of the potash deposits, it becomes a grave question as to how much salt they may be allowed to extract and bring to the surface.

In sinking a shaft that I was connected with, near Hanover, we had to go through 800 ft. of rock salt before we struck the potash deposit, which always, in that section, lies between what is known as the "Old Salt" and the "Young Salt." In this case the Young Salt was about 800 ft. in thickness, and we had to go through that. The question of what to do with that salt was a serious one. The government insisted that we build a covered bonded warehouse for it and all of the salt taken out of that 21-ft. diameter shaft for 800 ft. in depth had to be stored in the warehouse and kept in bond for the government. The suggestion that we should merely stack the salt up was not allowed, because in rainy weather the washings from it might probably be collected by the farmers

around and used for pickling, and so the government would lose the royalty on it.

Of course, after we got down into the potash deposits the trouble was not as great, and to-day those bonded warehouses are all empty. The salt has been taken back into the mine and used for filling where the potash has been taken out. That Dr. Day should have pointed out the real difficulty of potash recovery is very interesting.

Another matter in this paper which attracts my attention is the difference of the methods used in this country for the concentration of the brines from those used in Germany, Austria, and Hungary. Instead of using "aprons," as described in the paper, they erect a long staging, about 30 ft. high and varying from one to two or three miles in length. Both sides of the staging are filled in with brushwood, with the fine ends down. The brine pumped from the wells is conveyed along a perforated trough the whole length of these evaporating and concentrating stacks, and trickles down on both sides of the 30-ft. high brush heaps, and in that way the evaporation is carried on very cheaply, profitably, and perfectly. The droppings of concentrated brine are collected in troughs in the bottom, and conveyed to the salt-boiling establishments. It seems to me that in this country, particularly in the West, an adaptation of this method would be much cheaper than the methods now used, the evaporation in ponds, which necessitates the plowing up of the salt and the rehandling—when the concentration can be brought to the actual point of saturation by such a cheap and labor-saving contrivance as they have in Germany.

W. Lindgren, Boston, Mass.—I simply want to call attention to the fact that in the German schools a great deal more time is given to the subject of the treatment of salt than in the United States. You will recall, all of you, the title of the Prussian journal, *Zeitschrift für Berg-Hütten und Salinenwesen*, which is studied extensively, and in all the technical mining schools there are special courses on this subject.

David T. Day.—It seems to me that Mr. Spilsbury's proposition is one worth a good deal of study and consideration. The lifting of all this salt while it is in solution, by a simple pumping operation, means avoiding all shoveling, and manual labor, which is pretty hard to get in the neighborhood of San Francisco. To take this salt up in a saturated state seems to be an ingenious idea that ought to be developed.

E. G. Spilsbury.—The question of pumping the entire amount of sea water up to 30 ft. and distributing it as described would not be as economical as the first operations in the present salt pits or tanks. In Germany, their method has been developed, not for handling sea salt, but for pumping brine from wells. The brine is pumped from wells, and therefore the extra 30 ft. of pumping did not amount to very much. If it were applied here to the American conditions, I think it would be

more economical to make the first concentration in the solar pits and then pump the fairly concentrated brine up and finish the concentration on the brush evaporators.

H. Ries, Ithaca, N. Y.—Is that German method still as extensively used in Germany as heretofore?

E. G. Spilsbury.—Yes, you can notice these long high brush heaps for miles along the railroads, in the salt-well country.

H. Ries.—Does the bittern rejected at San Francisco bay contain any bromine?

David T. Day.—Yes, the amount of bromine seems to be considerable, but practically no use is made of it, except an extremely small amount for medicinal purposes.

Asbestos in Southern Quebec

BY JOHN A. DRESSER, SAULT STE. MARIE, ONT., CANADA

(Pittsburgh Meeting, October, 1914)

General

THE controlling supply of asbestos for the world is obtained from southern Quebec, 150 miles or less north of the international boundary line between Canada and the United States, and about 75 miles south of the city of Quebec. The principal production is furnished by eight mines, seven of which occur within a distance of 6 miles, but there are also several smaller properties in the vicinity. The industry was begun in a small way some 35 years ago and has advanced more or less regularly ever since. The annual production now exceeds 100,000 tons and its value is about $3,000,000. It represents over 80 per cent. of the world's production.

History

Asbestos has been known in the eastern townships of Quebec since 1847, when attention was called to it in an official report by Sir William Logan, the first Director of the Geological Survey of Canada, but it was not until 30 years later that it came into commercial importance. The largest deposits, those of Thetford and Black Lake, were found in 1877 during the construction of the Quebec Central Railway. Work was begun upon them almost at once and has been continued ever since.

The Danville mine, the next largest producer, was opened in 1879, and the slip-fiber deposits of East Broughton were located shortly afterward.

For the first 15 years only the "crude" asbestos was recovered; that is, fiber long enough to be extracted by hand cobbing. Although this is still a valuable part of the production, it is now a relatively small part of the total output.

After several trials, a process of mechanical concentration was begun about 1893 by some of the pioneer operators of the district, which with many modifications has been successfully used ever since. Although there have been numberless changes in the operation and appliances, the present practice is a direct development of the first principles of the earliest attempts at concentration, and much credit is due to those who

originated it. Its effect may be realized when it is stated that in leading mines to-day 95 per cent. of the quantity and 75 per cent. of the value of their total production is obtained by mechanical concentration.

The growth of the industry is best shown by quoting the production of a few years taken at regular intervals:

	Production, Tons	Value
1878	50	.....
1882	810	$52,650
1892	6,082	390,462
1902	30,219	1,126,688
1912	111,175	3,059,084

Access and Location

All the mines have easy railway access. The principal shipping stations are Thetford Mines, Black Lake, and East Broughton on the Quebec Central Railway, a part of the Canadian Pacific system, and Danville, on the Grand Trunk Railway. Thetford Mines is about 76 miles from Quebec, 67 miles from Sherbrooke, and 168 miles from Montreal. The other stations mentioned are from 4 to 18 miles from Thetford Mines, except Danville, which is some 50 miles distant, on the Grand Trunk Railway, 88 miles from Montreal and 86 miles from Quebec.

Geology[1]

The asbestos deposits are found in the hilly country of southern Quebec known as the Eastern Townships. Much of the district has been settled for upward of 100 years and is now generally occupied by small dairy farms. The hills are a continuation of the Green mountains of Vermont, a part of the Appalachian system.

The geological structure is complex. There has been intense folding, faulting, and regional metamorphism. Glacial drift conceals a great part of the rock surface, but glacial erosion has been an important factor in uncovering and exposing the deep-seated rocks in which asbestos occurs. The asbestos is in a series of basic igneous rocks which occur in stocks and sills, that have intruded sedimentary strata of Cambrian, Ordovician, and in places of Silurian age. These basic intrusives are part of the well-known and extensive series which appears at frequent intervals in the Appalachians from Georgia to Newfoundland, and yet in a distance of 2,000 miles the commercial production of asbestos is almost entirely limited to this small district.

[1] The geology and general character of the district are more fully described by the writer in *Memoir No. 22, Geological Survey of Canada* (Ottawa, 1913).

The principal types of igneous rocks are peridotite, pyroxenite, gabbro, and diabase. They are products of differentiation from a single magma, and are characteristically arranged in the order given above from the base upward in sills, and from the center outward in stocks. This order, it will be noted, is that of decreasing basicity and density. A relatively small amount of hornblende granite which is also present has generally been intruded a little later than the basic rocks.

Peridotite is altered in important amount to serpentine, and pyroxenite generally to soapstone, but in places probably also to serpentine.

Character and Mode of Occurrence

The asbestos is entirely of the chrysotile variety, and has essentially the same chemical composition as serpentine, in which rock only it occurs. There are two types of asbestos in this district: namely, "cross-fiber" asbestos, and "slip," "parallel," or "mass fiber," asbestos, the three terms being used almost interchangeably.

Cross-fiber Deposits.—This variety furnishes the major production, both in quantity and value. It occurs in veins up to 2 1/2 in., or rarely 3 in. wide; the greater number are 1/2 in. or less in width. The fibers, as the name implies, lie crosswise the vein.

The veins rarely reach a length of 200 ft., but are usually very much shorter, the greater number being only a few feet in length. They run in all directions through the rock, in places cutting one another abruptly, but more frequently uniting at meeting. A careful examination shows that many of the larger and more persistent veins show an approach to a rectangular arrangement, and probably represent joints in the primary rock. Others have a roughly parallel order and denote fractures due to regional compression, while many smaller veins truncate the corners of rectangular joint blocks in shell-like form.

Vein Structure and Origin.—The veins are usually divided into two parts by a thin seam of iron ore, generally magnetite, which is parallel to the sides and near the center of the vein. Bordering the veins on each side there is invariably a band of serpentine about three times the width of the vein. The country rock near the veins is peridotite of the most basic phase (dunite) that occurs in the district. There is incipient serpentinization in all parts of the peridotite, but commercial asbestos is found only between walls of completely serpentinized rock whose thickness is proportional to the width of the vein. Microscopic and chemical evidences, as well as the distribution of the veins, point conclusively to the origin of the asbestos by alteration and recrystallization of the country rock *in situ*. Microscopically it is evident that the asbestos fibers have grown outward on each side from the seam of iron ore mentioned above; chemically

there is no distinction between the asbestos and serpentine, while an essential difference between serpentine and peridotite is that the former contains 12 to 14 per cent. of water. The position of the veins and the deep-seated character of the rock make it impossible that such open fissures ever existed where the veins are now found. While much remains to be explained both as to the processes and their causes, the facts are self-evident that zones of the country rock have been altered to serpentine and proportionate parts of these have taken the form of asbestos veins. From the persistence of the seams of iron ore within, or at the sides of, the veins, and the correspondence of the position of the principal veins to that of joint and other fractures, the iron seams are believed to mark the original channels by which circulating waters have been introduced which produced the hydration of the peridotite and its alteration to serpentine, a certain portion of which has crystallized as asbestos.

The source of the water has given rise to considerable discussion. The fact that much excellent asbestos has been found near granite dikes has given rise to the opinion that magmatic waters accompanying the intrusion of these has been the cause. Locally there is much to support this view, but viewed broadly the small amount of granite as well as its variable distribution make it improbable that it has been the chief cause. Magmatic waters accompanying the peridotite itself are a more likely cause. Yet, this rock was solidified and was afterward fractured both by contraction (joints) and by regional deformation before the waters could have been introduced. On the other hand, the present water level is comparatively near the surface in this district, and, as the region is one that suffered great erosion in glacial times, the present surface may long have stood below ground-water level. It is not proved that meteoric waters are not a competent cause to produce these effects and at least a share in the cause may probably be attributed to them.

In the deepest workings yet made, which are a little more than 200 ft. deep, there is no apparent change in the quantity or quality of the product, and no deposit of serious importance has yet been exhausted. The principal deposits are found in the most basic parts of the peridotite (dunite); that is, near the base of sills or toward the central parts of stocks. Owing to glacial erosion having been more effective on the north side of the stocks several deposits are found near the north edge of the peridotite core of the principal stock.

Slip-Fiber Deposits.—Slip-fiber or parallel-fiber asbestos is a fibrous phase of serpentine in which the fibers are arranged parallel to adjacent cleavage faces in the rock. In places the rock is almost entirely in a fibrous condition and the name "mass-fiber" is sometimes used for such occurrences. It is probable that this class of asbestos has been derived from pyroxene.

Where slip fiber is abundant the proportion of asbestos in the rock is

much higher than in the case of the cross-fiber occurrences. But much of the slip fiber is short and the proportion that is recoverable and useful, though higher, is not so greatly different from that obtained from the cross-fiber mines. Only a little "crude" asbestos is obtained from deposits of this class.

Mining and Dressing[2]

Mining.—The distribution of the asbestos is such that all the rock within the area mined must be handled. Except in one mine where underground work is quite extensively carried on, though principally in winter, or for development purposes, the mining, or rather quarrying, is all open-cast work. The ground is cut down in benches, generally 6 to 12 ft. high, which are carried across the floor of the pit so as to afford sufficient working face. Several of the pits have reached a depth of 200 ft. from the original surface and are from 600 to 1,200 ft. or even more in length.

Handling.—Hoisting is done by means of cable derricks with boxes carrying about a ton each. In one case a tramway enters the pit through an inclined tunnel and the rock is hauled out in cars drawn by a cable. Hauling on the surface is done by small locomotives with side-dumping cars of 4 or 5 tons capacity.

Dressing.—The separation of fiber from the rock commonly begins in the pit. Rock containing "crude"—that is, veins 3/4 in. or more in width and of good quality—is sent to the cobbing sheds for hand separation, and asbestos that is liberated by the breaking of the rock in the pit is collected in hand boxes; dead rock is taken to the waste dump, and the remainder, usually 35 to 60 per cent. of all the rock handled, goes to the ore bins, or directly to the mill for mechanical concentration.

The milling practice varies somewhat in different mills, but is very similar in all. It consists essentially of coarse crushing, drying, and alternate finer crushings and screenings. At each screening, the asbestos then liberated is drawn off through overhead pipes by suction fans and collected in settling tanks. When thoroughly screened from dust and classified according to length of fiber by means of a rotary screen, the different grades pass to their respective storage bins, or in some mills is mechanically bagged.

In the coarse crushing, jaw crushers are used, and gyratories and frequently rolls for the finer crushing. When rolls are used special appliances are needed for teasing out the fiber, which becomes compressed into matted sheets by the rolls. The final crushing of the rock is effected by a specially designed "cyclone." This consists of two "beaters" or fans of chilled iron, in shape like the screw propeller of a boat and weighing

[2] The methods of mining and concentrating have been very fully described by the late F. Cirkel in *Publication No.* 69, *Mines Branch, Department of Mines* (Ottawa, 1910).

upward of 100 lb., which revolve at a speed of 2,000 rev. per minute, or more, in a closed chamber. From the rock fragments thus driven together the smallest particles of asbestos are released and collected as before.

Suction fans for the removal of dust from the cyclone, the classifier, and sometimes from the mill, are important accessories to the equipment. Magnets are usually placed over the shaking screens to eliminate particles of iron ore. The average recovery of cross fiber seems to vary at different properties between 3 and 8 per cent. of the rock treated; slip fiber, perhaps 2 or 3 per cent. higher.

Product.—The fiber recovered in the mill is classified into three or more grades, the crude asbestos usually into two grades. The adoption of a standard classification has been discussed, but owing to local differences in the character of the fiber, as well as other causes, no standardization has yet been effected. Each mine follows its own grading and there is a lack of uniformity in the products of different mines. An arbitrary classification that has been adopted by the Department of Mines of the Province of Quebec is as follows:

Crude Asbestos, Hand Cobbed

No. 1.—Value $200 per ton or more.
No. 2.—Value less than $200 per ton.

Mill Stock, Mechanically Separated

No. 1.—Value $45 per ton or more.
No. 2.—Value $20 to $45 per ton.
No. 3.—Value less than $20 per ton.

The production of 1912 was as follows, according to the classification given above:

Quantity of Rock Mined 1,870,608 Tons			Shipments			Stock on Hand	
Qualities	No. of Men Employed	Wages Paid	Tons	Value	Av. per Ton	Tons	Value
Crude No. 1...	.....	.....	1,914	$510,785	$263.16	867	$221,215
Crude No. 2...	.....	.....	3,766	379,445	100.76	2,867	310,596
Mill Sk. No. 1.	.....	.....	3,682	237,203	64.42	2,370	137,106
Mill Sk. No. 2.	.....	.....	32,689	1,018,960	31.17	8,234	301,774
Mill Sk. No. 3.	.....	.....	69,097	912,691	13.21	6,838	131,206
Totals.....	2,910	$1,377,444	111,175	3,059,084	27.52	24,176	1,102,206

The stock on hand at the beginning of the year 1912 amounted to 33,751 tons, valued at $1,583,076.

Uses.—A small proportion of the crude asbestos is used for making asbestos cloth and various fire-proof textiles. A much greater amount

is used for covering and insulating purposes. Boards, shingles, and roofing felts for fire-proof construction, materials for electric insulation and protection from acids, and boiler and pipe coverings are the forms of manufacture in most common use. There is only one plant for the manufacture of asbestos goods in Canada, the Asbestos Manufacturing Co., at Lachine, Quebec. This plant makes shingles and other roofing materials, pipe covering, mill boards, and asbestos lumber for the Canadian and European markets. Its consumption is about 5 per cent. of the total output of the mines of Quebec. Approximately 75 per cent. of the output of these mines is exported for manufacture to the United States, and the balance to Great Britain, Germany, France, and other European countries.

Condition and Outlook for the Industry

There has been a decrease of stock on hand during the past two years. This seems to indicate that the industry is emerging successfully from a critical condition into which it was thrown four years ago by a rapid overproduction, which resulted from some extravagant financial promotions.

The deposits are large and the principal mines seem to have almost inexhaustible reserves. Consequently the plants installed are of a durable character and around the mines substantially built towns have grown up. The labor is largly obtained in the locality. Many of the men own their houses, living conditions are favorable, and there have been no serious labor troubles in the history of the industry.

For the immediate future, or as long as mining can be done by open-cast methods, the cost will probably vary only with the nature of the ground, the depth of hoisting, and the price of labor and materials. But in methods of concentrating changes are more likely to take place. The present practice is the result of 20 years' experience, during which time many changes and great improvements have been made. At present the practice of different mills varies considerably in details and probably also in efficiency.

The enlargement of the market for manufactured goods, which is steadily growing, depends largely on the skill and enterprise of the manufacturers and the supply of substitutes for asbestos. The constantly increasing price of lumber must improve the field for shingles and asbestos lumber, while the ingenious applications of asbestos for heat-resisting and insulating materials seem likely to give it a continued advantage over any substitutes at present known.

The absence of rival fields for so long a time is rather remarkable. Similar rocks in like associations are known in many parts of the world, yet the only sustained and growing production outside of Quebec is obtained from the Ural districts in Russia, the United States and South Africa

being much smaller producers. In his report on Mining Operations in the Province of Quebec for 1912, T. C. Denis, Superintendent of Mines of Quebec, cites the following statistics for the year 1910, from the Colonial and Foreign Statistics of the Home Office, London. The quantities are given in metric tons and values in pounds sterling:

	Production Metric Tons	Value
Cape Colony	1,273	£23,143
Cyprus	442	2,754
India	3	6
Rhodesia	301	3,320
Natal	2	15
Transvaal	70	2,575
Russia (approximate)	10,936	82,000
United States	3,350	14,036
Total outside of Quebec	16,377	£127,849
Quebec	73,124	£548,184

Discussion

E. G. Spilsbury, New York, N. Y.—Referring to the methods of crushing, is there any special way of feeding the material—the cross-grained asbestos—to the rolls so that the crushing effect is only lengthways of the fiber, or does it all have to go in and produce its portion of broken fiber?

John A. Dresser, Sault Ste. Marie, Ont.—Mr. Spilsbury's question is a very important one, economically. Unfortunately, it is not possible to feed the crushers in any way such that the fiber may not be broken and shortened, and its value much reduced. The coarse crushing is usually done by jaw crushers, after which gyratory crushers are used for the finer crushing. Rolls are also used for the finer crushing. Where rolls are used it is necessary to have a teasing apparatus for bringing out the fiber, which is apt to become matted by the action of the rolls. Any hard material in the rolls is apt to break and shorten the fiber. If we have asbestos fiber $\frac{5}{8}$ in. long that might be worth $100 a ton, and that fiber is broken into two parts, it might only be worth about $30 to $40 a ton. Some mills have attempted to do without rolls entirely and are doing so. After the fiber to about $\frac{1}{4}$ in. has been removed, the material is fed to an especially designed machine known as the "cyclone" crusher. It is said to be a development of the principle of the fanning mills used in the West for blowing the husks off grain. It consists of steel chambers in which are two fans made like screw propellers, which usually face each other and are carried on horizontal shafts. These weigh 90 to 100 lb., and revolve at about 2,000 rev. per minute, in opposite directions, and

the material fed in is finely pulverized. The smallest particles of asbestos are thus released. When the material reaches that stage it means almost the destruction of the fiber; so the practice is to draw off the fiber wherever it is possible to do so. But there seems to be no means of doing this completely without seriously lessening the value of the product by shortening the fiber.

DAVID T. DAY, Washington, D. C.—What is the material known as asbestic?

JOHN A. DRESSER.—An ordinary mortar carrying a certain percentage of very short fiber, used as a binder, and possibly because it has a little better fire-resisting qualities than ordinary mortar. The fine material which is added to the lime is hardly distinguishable from dust. It is said to give slightly better fire-resisting qualities, but I do not know that its binding qualities are especially important. It gives a very smooth surface. It is a by-product at one of the larger mines.

D. T. DAY.—Is there any effort made to use the serpentine which invariably occurs on either side of the cross-fiber vein?

J. A. DRESSER.—No, except in so far as the tailings are used for asbestic.

D. T. DAY.—What has become of the deposit of asbestos in the bottom of the Bright Angel trail in the Grand Canyon.

W. LINDGREN, Boston, Mass.—It is still there, I have no doubt, but the question is to get it away from there.

I have a couple of questions I would like to ask Mr. Dresser. We all know what exhaustive examinations he has made in the asbestos regions, so I would be glad to have him give the results of his examination regarding these queries: In speaking of the slip-fiber variety of asbestos, does the transformation of the rock to this material go through the serpentine stage or is it directly altered to asbestos from pyroxene or amphibole? Another question concerns the gravitative differentiation of stocks into acidic and basic rocks. We know that such separation takes place in sills, but is it not a little difficult to apply this theory to stocks? In stocks the separation usually takes place along the periphery, the basic material seemingly consolidating first along the margins.

J. A. DRESSER.—In so far as is known from a general examination, I think that the alteration in the first instance has been to serpentine in each case. In the case of slip-fiber asbestos deposits, however, the serpentine is principally derived from pyroxene, but this pyroxene is probably primary.

As to the question of differentiation of stocks, there is a variation vertically as well as horizontally. We find stocks that have an entire

ring-like arrangement, though they are seldom symmetrical. The basic core is frequently found near one side of the stock in this annular arrangement. We get this in all states from a complete annular arrangement to an arrangement going three-fourths way around and finally passing out into a laccolith, or thickened sill. Differentiation seems to have taken place within the stocks themselves as there is an acid cap forming the upper parts of many of them. So we have cases of hills showing serpentine near the base but covered entirely over the top by diabase or porphyrite.

In one case a man, who thought that mining meant always driving a drift or shaft, drifted into diabase at the side of a hill, passed through diabase, pyroxenite, and finally got into peridotite, in a distance of scarcely 100 ft. Peridotite, pyroxenite, and diabase are all differentiations from a single magma and arranged in this order from the base upward and from the center outward. This arrangement, which is well established for this district, is comparable to that described recently by Duparc and Pamfils in the Ural district in Russia.

H. A. Wheeler, St. Louis, Mo.—Some years ago a deposit of asbestos was discovered in the serpentine hills at Tompkinsville, Staten Island, N. Y. It occurred in fibers 10 to 20 in. in length, that were very coarse and brittle. After some desultory prospecting it was finally abandoned, as it occurred in small pockets, not veins.

H. Ries.—What is the greatest depth recorded at which asbestos is found? You said pits had reached the depth of 200 ft., I believe.

J. A. Dresser.—As far as I know 325 ft. is the maximum depth of the largest pit. Borings are reported from which some little fiber was obtained at 400 ft.; but I do not know in what quantity. There appears to be little difference in the quality at different depths. It is always irregularly distributed, making the ground extremely spotted. Because we have it at 300 ft. in some places it does not necessarily occur at that depth in another place. I should say that is the greatest depth that it is undoubtedly proved. We are usually able to work the pits all winter. At one plant there are some underground workings of considerable extent. Ordinarily there is a good deal of loss from snowfall if the output is not properly managed, but under ordinary market conditions all can be worked the year around. The rain causes a serious hindrance in workings, as well as snow.

Asbestos Deposits of Georgia*

BY OLIVER B. HOPKINS,† ATLANTA, GA.

(Pittsburgh Meeting, October, 1914)

As prefatory to the body of this paper, a few general statements will be made (1) in regard to the history and importance of the asbestos industry, (2) as to the principal sources of the raw material, and (3) as to the types and modes of occurrence of asbestos.

While the modern asbestos industry is comparatively new and its growth during the last decade or two has been marvelous, asbestos has been known and used in a small way for probably 2,000 years—at least since the time of the Roman Empire. Pliny refers to it as "the funeral dress of kings" and Plutarch records its use as lamp wicks. In the middle of the 18th century the deposits of the Ural mountains were opened up and the first factory for the manufacture of asbestos goods was established. But, owing to the limited demand for the goods, the industry disappeared and interest lapsed until about 50 years ago. Since 1860, the asbestos industry has made wonderful strides, and each year sees new uses to which the material can be put, and hence an increased demand and an increased output.

Of the three great producers of asbestos, Canada, Russia, and the United States, Canada stands pre-eminent, producing from one-half to three-fourths of the world's supply; Russia comes second, producing in 1911 about two and one-half times as much as the United States. In 1912 the Canadian production,[1] including both asbestos and asbestic, was 131,260 short tons, valued at $2,979,384; while during the same year the production of the United States[2] was 4,403 tons, valued at $87,959. Canada holds the first place in the production of raw asbestos, while the United States holds the first place in its manufacture.

Although the production of asbestos in the United States is comparatively small there are deposits of varying importance in the following States: Vermont, Virginia, North Carolina, Georgia, Wyoming, Idaho,

* Published by permission of the Geological Survey of Georgia.

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[1] *Summary Report of the Mines Branch of the Department of Mines of Canada*, year ending Dec. 31, 1912.

[2] J. S. Diller: *Mineral Resources of the United States*, 1912, Advance Chapter on Asbestos.

Arizona, and California. Of these States, asbestos has been worked commercially in Vermont, Virginia, Georgia, and to a small extent in Wyoming, Idaho, and California. In production, Vermont ranks first and Georgia second.

Asbestos-Forming Minerals

The asbestos-forming minerals come naturally under two heads, amphibole and serpentine. In the amphibole group there are a number of different minerals of quite variable compositions, while under serpentine there are only two which have practically the same composition, and only one which is of importance. The following tabulation shows the relationships of these minerals:

Asbestos-Forming Minerals:
Amphibole Group.
Orthorhombic division.
Anthophyllite, $(MgFe)SiO_3$.
Monoclinic division.
Tremolite, $CaMg_3(SiO_3)_4$.
Actinolite, $Ca(MgFe_3)(SiO_3)_4$.
Mountain Leather,
Mountain Wood.
Mountain Cork.
Crocidolite, $NaFeSi_2O_6.FeSiO_3$.
Serpentine, $H_4Mg_3Si_2O_9$.
Var. Chrysotile.
Var. Picrolite.

The most important of these minerals in point of view of quantity and accessibility are chrysotile, anthophyllite, and crocidolite; while from the point of view of commercial application chrysotile is by far the most important with anthophyllite probably second. The amphibole type of asbestos occurs in Georgia, Idaho, North Carolina, and Virginia; and the serpentine variety occurs in Vermont, Wyoming, Arizona, and California.

What are the properties of asbestos upon which its value depends? An answer to this question involves the knowledge of the physical properties of asbestos and gives an insight into the uses to which the material is put. The most important properties are its fibrous quality, resistance to heat and acids, and low conductivity. Under the head of fibrous quality the following characteristics may be considered: Fineness, length and flexibility of fiber, and tensile strength. Not only do the various types of asbestos vary widely in regard to these properties, but the relative value of the properties varies with the use to which the material is put. In regard to fineness, length, flexibility of fiber, and tensile strength, crocidolite is quite the equal of chrysotile, and superior in some respects; but in fire-resisting properties crocidolite is seriously lacking. Anthophyllite is equal to any other type of asbestos in regard

to its resistance to acid and heat and insulating properties, but far inferior to chrysotile in regard to flexibility, fineness of fiber, and tensile strength. Thus it is that the anthophyllite, and the related types of amphibole asbestos, which constitute the most important deposits in the Southern States, are classed as low grade. Anthophyllite is not suitable for some of the purposes for which chrysotile is used, and hence it is considered of little or no value by some of the producers of higher grade material; but where quality of fiber is not important and where resistance to heat and acids and low conductivity are essential, anthophyllite is as valuable as any other type of asbestos.

Asbestos occurs in three forms:[3] as cross fiber, slip fiber, and mass fiber. When the asbestos occurs in veins with the fibers at right angles to the inclosing walls it is called cross fiber; when the fibers are parallel to the inclosing walls it is called slip fiber; and when the asbestos forms the entire rock mass composed of bundles of fibers arranged in all directions in respect to each other, often in somewhat radiating forms, it is called mass fiber. Cross fiber may be converted to slip fiber by movements of one wall in respect to the other. Slip fiber is found in the slipping plane, and the direction of the fibers records the direction of motion.

Chrysotile occurs as cross and slip fiber, but never as mass fiber; its characteristic mode of occurrence is as cross fiber. Fibrous tremolite and actinolite occur only in slip-fiber veins, while anthophyllite occurs in all three forms, but its characteristic form is as mass fiber.

Diller[4] has recognized four modes of occurrence of asbestos in the United States, which are as follows:

"The first mode is as cross-fiber veins of chrysotile in serpentine derived from peridotite, a deep-seated igneous rock, as near Lowell, Vt., and Casper, Wyo. So far as known, it is much the most important mode of occurrence and is well illustrated at the Thetford mines, in Canada.

"A second mode is as cross-fiber veins of chrysotile with serpentine in limestone. Its most important illustration is in the Grand Canyon of the Colorado in Arizona.

"The third mode of occurrence is as mass-fiber amphibole (anthophyllite), composing stocks and dikes of fibrous amphibolite, and is well illustrated in the deposits worked for many years at Sall Mountain, Ga.

"The fourth mode is as slip-fiber veins in rocks which for the most part are cortlandite and pyroxenite, but which locally pass into peridotite, as at Bedford and Rockymount, Va."

Location

The asbestos deposits of Georgia are associated with the crystalline schists of pre-Cambrian age which occupy both the Piedmont Plateau and the Appalachian Mountain provinces. While asbestos has been

[3] Diller, J. S.: *Bulletin No.* 470, *U. S. Geological Survey*, p. 506 (1911).
[4] *Op. cit.*, pp. 506, 507.

found associated with basic igneous rocks over a large part of the area of crystalline schists, the distribution of commercial deposits is much more limited, and is, in general, confined to a belt extending from Rabun and Towns counties southwest into Alabama in the neighborhood of West Point. And even in this belt, which includes all of the deposits likely to be of commercial value, the most important deposits are grouped in its northern part, and more specifically in Rabun, White, and Habersham counties.

Associated Rocks

The asbestos deposits of Georgia are limited in their distribution to the areas of basic, igneous rocks, which are represented in large part by the following types: Hornblende schist, hornblende gneiss, diorite, gabbro, peridotite, and pyroxenite. All these types of rocks, except peridotite and pyroxenite, have a wide distribution in this State. The hornblende schist and hornblende gneiss are found in varying quantities in every county in the crystalline area, occupying long narrow bands and folded with the rocks of the Carolina gneiss series, which vary from mica schist to granite gneiss. The gabbros and diorites are found associated with the hornblende rocks, into which they sometimes grade, occupying broader and less elongated areas.

Associated with these hornblendic rocks, diorites, and gabbros are small deposits of slip-fiber asbestos, which give little or no promise of commercial value, and the distribution of these deposits is practically coextensive with the distribution of those rocks.

The peridotites and pyroxenites, on the other hand, are more limited in their distribution and it is with them that the most important asbestos deposits of the State are associated. Their distribution is, in general, limited to the belt in which the most important asbestos deposits of the State occur.

Types of Asbestos Found

Asbestos has been found in Georgia representing three of the four modes of occurrence described by Diller as occurring in the United States, namely: Chrysotile in cross-fiber veins associated with serpentine, which is derived largely from peridotite; mass-fiber anthophyllite in dikes or irregular masses; and slip-fiber veins in different types of basic rocks, varying from courtlandite to peridotite or pyroxenite.

Asbestos of the first type, as chrysotile in serpentine rocks of igneous origin, is found in Georgia in a limited number of places and in even more limited quantities. At none of the places examined were the veins more than $\frac{1}{8}$ to $\frac{1}{10}$ in. in thickness; the veins were very limited in distribution, and constituted a very small portion of the rock mass. It

can be quite definitely stated that this type of material does not occur in commercial quantities in this State.

Amphibole asbestos is of much more general distribution and is present in much larger quantities. Slip fiber occurs at numerous localities over the crystalline area of the State, where it is found along shearing planes in the basic rocks described above. Its commercial possibilities, while somewhat more promising than the deposits of the chrysotile type, are as a rule unworthy of consideration because of the large amount of waste material which has to be removed, since the veins are usually not more than a few inches in thickness and seldom reach as much as 8 to 10 in., although they are exceptionally 16 to 18 in., in width.

Thus the only type left for consideration is the mass-fiber anthophyllite deposits. These deposits are unique in that more than 90 per cent., usually as much as 95 per cent., of the rock quarried is realized as fiber, while in the case of the chrysotile deposits of Canada the average extraction of fiber for 1912 was only 6.45 per cent. of the rock quarried. Although the extraction of the Canadian chrysotile is only about one-fourteenth that of the Georgia mass-fiber asbestos, the price is less than three times as great. This gives a decided advantage to the Georgia product, especially since the mining can be done more economically and the milling is much more simple.

Canadian Asbestos[5]	Georgia Asbestos
Extraction, 6.45 per cent.	90 to 95 per cent.
Average price, 1912, $27.79 per ton	$10 to $12 per ton.[6]

Extraction, one-fourteenth that of the Georgia anthophyllite.

Price, two and four-fifths times that of Georgia anthophyllite.

The circumstances which keep the Georgia deposits from being developed are the limited demand for the material, the distance from the railroad, and the limited size of the deposits, features which will be dealt with later.

Description of Localities

A detailed description of the more important asbestos deposits or even an enumeration of them would serve no useful purpose in this connection; however, a brief account of the best-known deposits, as typical of the class, may serve as a general picture of them all. Figs. 1 and 2 are views on the property of the Sall Mountain Asbestos Mfg. Co. and Fig. 3 shows a deposit at the mine of the Asbestos Mining & Mfg. Co., Hollywood, Ga.

[5] *Summary Report of the Mines Branch of the Department of Mines of Canada*, Year ending Dec. 31, 1912, p. 156.

[6] The figures here given represent the general range in value for several years and not the value for 1912, as that information is confidential, there being only one producer.

Sall Mountain Mine

The Sall Mountain mine is the oldest producing asbestos mine in the United States. It began operation about 1894, and until the last few years it was the largest producer in the United States. The mill has a capacity of 10 tons per day. It is the only mine, except the one at Kamiah, Idaho, which was worked a few years ago, where mass-fiber anthophyllite has been mined.

The rock is a fibrous amphibolite in which the bundles of fibers range from $1\frac{1}{4}$ in. down to a small fraction of an inch in length. But, owing to the lack of flexibility, the fibers break into short lengths on being

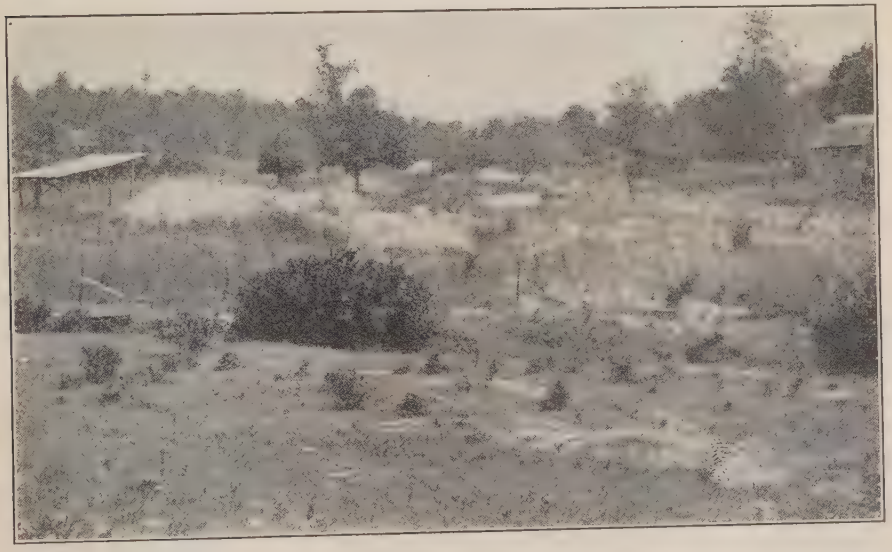


FIG. 1.—WEATHERED ASBESTOS ROCK DUG FROM SMALL PITS ON SALL MOUNTAIN COMPANY PROPERTY NEAR CLEVELAND, GA.

fiberized, so that the longest fiber is about $\frac{1}{4}$ in., and ranges down to $\frac{1}{10}$ in. and less. More than 95 per cent. of the rock mass is converted into fiber. Pyrite, magnetite, talc, and carbonate of lime and magnesia constitute the impurities. The asbestos is grayish white when fresh, but a large part of that produced has been stained yellow by surface water.

The country rock is a gneiss cut by more recent granite. The asbestos occurs in lens-shaped masses, of which six of varying size have been worked in an area of not more than 20 acres. The largest body was about 75 ft. long by 50 ft. wide as a maximum, and has been worked in an open cut to a depth of more than 50 ft. This company has largely worked out its deposits on Sall mountain, but owns others near Asbestos station on the Gainesville & Northwestern Railroad, where the material

FIG. 2.—ASBESTOS MASS BENEATH DECOMPOSED GNEISS, SALL MOUNTAIN MINE.

FIG. 3.—ASBESTOS ROCK ON THE PROPERTY OF THE ASBESTOS MINING & MFG. CO., HOLLYWOOD, GA.

is identical in association and character, except in the amount of disintegration which surface waters have produced.

Mass-fiber anthophyllite occurs at a number of other localities in White, Habersham, and Rabun counties, usually associated with partly altered rock, slip fiber, and talcose material. There are a number of places which give promise of yielding as much or more asbestos rock than the Sall Mountain property. The occurrence at all these places is very similar and the origin appears to be the same, but the quantity of asbestos at an individual place is difficult of determination because of the irregular shape of the bodies and the variation in the amount of alteration in the individual mass.

Origin of Mass-Fiber Deposits

From the field relationships and the microscopic study of much of the material from Georgia the conclusion has been reached that mass fiber develops from olivine-enstatite rocks.

In regard to field relationship, it has already been stated that the important asbestos deposits, those of mass fiber, are limited to the belt of peridotites and pyroxenites. Wherever there is any rock associated with the asbestos which shows an original mineral or minerals it is universally found to contain olivine or enstatite, or both.

A microscopic study of the rocks in thin section has developed the facts that olivine and enstatite give rise to anthophyllite, and that all gradations from olivine-enstatite rock to anthophyllite are present. The alterations noted are briefly as follows: The development of slender needles of anthophyllite running in all directions in the olivine-enstatite rock and penetrating the crystals regardless of cleavage cracks and crystal boundaries; the increase in number and size of anthophyllite needles at the expense of the original minerals until only skeletons remain; and finally the anthophyllite rock is formed. But this alteration may have been stopped at any stage in its progress by changed physical conditions; by processes of hydration and oxidation, the anthophyllite, as well as the remaining olivine, may be partly converted into serpentine and the latter partly to talc; the result may be a rock containing olivine, anthophyllite, serpentine, talc, magnetite, and carbonate of calcium and magnesium.

Quarrying

The mining of asbestos in Georgia has been entirely from open cuts or quarries. Rough, brownish boulders usually outcrop at the surface. By a study of the distribution of surface fragments and by digging a few shallow pits the size of a deposit near the surface may be fairly

well outlined, although the irregular shape of the bodies precludes even a rough estimate of the amount of material present.

When work is started, the deposit is followed in all directions; but the removal of the rock is commonly begun along the contact with the country rock, as there the rock is usually soft and a face can be easily made to blast from. The removal of soil at all the localities where development has been undertaken is usually a small item. The rock is frequently soft enough to dig with a pick, although irregular masses of hard rock have to be drilled and blasted. The work is done largely by hand; the rock is loaded in small hand cars or wheelbarrows and hauled to the drying sheds.

Milling

The crude asbestos is dried either in open sheds or over steam pipes; when sufficiently dry it is passed through a jaw crusher, and then through a rotary crusher, where the rock is reduced to 1-in. mesh and less. From the crusher, the rock is passed to a bin, from which it feeds into a Raymond pulverizer. As the rock is fiberized it is carried upward by a suction fan and separated by air currents, the coarse material returning to the pulverizer, the fines collecting in a bin which feeds into a packer, and the dust settling in a dust room.

Uses of Asbestos

One of the principal uses to which the anthophyllite asbestos has been put is in the preparation of asbestos cement for boiler coverings and furnace linings. Twelve (?) car loads of the Georgia product were used at Ducktown in lining the flues. This type of asbestos may be used in the manufacture of the following articles: Furnace linings, fire bricks, steam-pipe coverings, asbestos shingles, wall plaster, asbestos tiling for floors, etc. Numerous experiments have been made with the amphibole asbestos, but, owing to the fact that the largest manufacturers of asbestos in this country are interested in the Canadian mines, it is difficult to get this material on the market in the United States. The greater part of the mineral now being mined in Georgia is sold in Germany. During the past year the manufacture of asbestos cement from Georgia asbestos was begun here and a considerable quantity has been sold for local use. Considering the uses to which it can be put and the cheapness of its extraction and milling, it seems that there should be a future for the material, notwithstanding the enormous output of low-grade asbestos from the chrysotile areas.

Summary

Asbestos representing three modes of occurrence is found in Georgia. Chrysotile, occurring in serpentine which is derived from peridotite,

is present in insignificant quantities in a few localities, where it gives no promise of commercial value. Asbestos of the amphibole variety in slip-fiber veins occurs at widely distributed points over the Piedmont area of the State; while mass-fiber asbestos, which represents the most important deposits from a commercial point of view, is restricted in general to the belt of peridotite and pyroxenites which crosses the State in a southwest direction from Rabun county to Harris county, but is relatively more important in Rabun and Habersham counties than in any others. Judging from the field relations and the microscopic study of the material it has been concluded that the mass-fiber anthophyllite is derived from enstatite-olivine rocks.

Mass-fiber asbestos, owing to the nature of its occurrence, is capable of being mined very economically, but owing to the slight demand little is being put on the market at the present time. With a good demand for the material, at from $8 to $12 per ton a number of deposits in this State could be worked with profit and a large amount of the material could be put on the market.

Quarrying Shale by the Tunnel System

BY DWIGHT T. FARNHAM, SEATTLE, WASH.

(Pittsburgh Meeting, October, 1914)

Description of Quarry

THE shale used at the Renton plant of the Denny-Renton Clay & Coal Co. for the manufacture of vitrified paving brick occurs in a hill rising from 200 to 300 ft. above the level of the valley in which the plant is situated. The shale deposit, which is of unknown thickness, pitches about 10° to the southeast. The 150-ft. face worked extends approximately east and west, so that the dip, as shown by the photograph, is to the left and into the hill. The shale is overlain with about 50 ft. of gravel, 70 ft. of clay, and 30 ft. of mixed loam and gravel, making altogether from 125 to 150 ft. of overburden.

Disposal of Overburden

The overburden is removed by sluicing. For this purpose a No. 2 Joshua Hendy giant is used, throwing a $1\frac{1}{2}$-in. stream, and having 60 lb. pressure at the nozzle. The water is drawn through an 8-in. pipe from Cedar river, about 1,000 ft. distant, by means of a Worthington four-stage centrifugal pump, direct connected with a 100-h.p. motor placed just south of the factory, and 1,500 ft. from the nozzle This plant handles the material satisfactorily to a height of about 75 ft. above the level of the top of the shale. When the higher levels are worked, an auxiliary plant consisting of a four-stage Byron Jackson centrifugal pump with a 100-h.p., belt-connected motor, is thrown into the system. This plant is situated on top of the shale and can be discerned at the extreme right of the trestle work in Fig. 1.

The overburden is worked in either two or three benches, depending upon the height of the hill at that particular point. Fifty feet is about the extreme height which can be worked at one time with any degree of efficiency with the plant.

The usual method of working is as follows: A cut is made about 50 ft. in height and 100 ft. in depth, extending completely across the face, which is about 800 ft. long at present. The material is shot down, men being lowered over the brink by ropes, when necessary, to do the drilling. The

clay, loam and gravel fall in large lumps, some of which need further reduction, depending upon the nature of the material.

The stream from the giant reduces these to a slurry containing lumps up to perhaps 4 in. in diameter. Once a cut is finished, the bank is sloped to an angle of about 45°, and the next cut below started and completed in the same manner until the shale is uncovered. Fig. 2 shows the stream playing on the 50-ft. gravel stratum which overlies the shale.

Sluicing operations are carried on continuously. The men work in three shifts, arc lights and 5,000-c.p. acetyline headlights being used at night. In this way 19,000 to 20,000 yd. of overburden are moved each month.

The material is removed from the face by means of ground sluices which connect with the boxes supported on the trestles shown in Fig. 1. These boxes are 10 by 12 in. inside, and the bottoms are lined with vitrified brick, which outlast any other material. When running gravel, which gives the hardest wear, iron boxes last 90 days, wood blocks six weeks, while the brick remain in service from 10 months to a year.

Gravel which is suitable only for ballast is run directly into railroad cars by means of a sluice box as shown in Fig. 1. To the left of this can be seen the gravel trestle which leads such gravel as is suitable for concrete work to a washing and screening plant. Enough gravel is sold to reduce the cost of removing the overburden 12 per cent. The trestle to the left carries the boxes which remove the clay and loam and other unsalable material. The larger lumps, at the time the photograph was taken, were settled behind the large dike just across the railroad tracks, while the finer material in the form of slurry was run underneath the tracks and settled in a series of basins.

Great care is taken to prevent any of this material from flowing into Cedar river, which is being made a navigable stream.

Large rocks which cannot be sluiced, are shot and allowed to remain on top of the shale until the shale is used, when they are taken out over the floor of the quarry.

Former Quarrying Methods

The quarry floor is perhaps 10 ft. above the level of the factory. The shale, which is used in the manufacture of vitrified paving brick at the rate of 700 tons per day at present, presents a face 700 ft. long and 150 ft. high. Various methods of working have been tried, but none proved so successful as the present one. A steam shovel is, of course, out of the question with so great a lift. Only about 30 ft. of this material could be worked efficiently by shovel, and the vision of five steam shovels working one above the other is too appalling to contemplate when only about 700 tons of shale a day is to be moved.

We did try, however, a ½-yd. orange-peel bucket and a locomotive crane. With this outfit an attempt was made to pick up the shale, once it had been shot down, and to drop it into cars which were dumped over a grizzly having bars spaced 6 in. apart, which allowed the fine material to fall directly into the conveyor, while the coarse material was led to a crusher. This proved unsatisfactory on account of the large blocks of shale, all of which had to be shot and re-shot. Furthermore, the bank is made up of alternate layers of sandy and pure shale, which require it to be

FIG. 1.—SHALE QUARRY AND

evenly mixed in order to produce the highest grade of brick. This outfit did not allow proper mixing, and on account of the lumps, had a capacity of less than 300 tons per day.

After this experiment, we reluctantly went back to our old system of shooting the face as widely as necessary, and then gathering the material with shovels and by hand into cars.

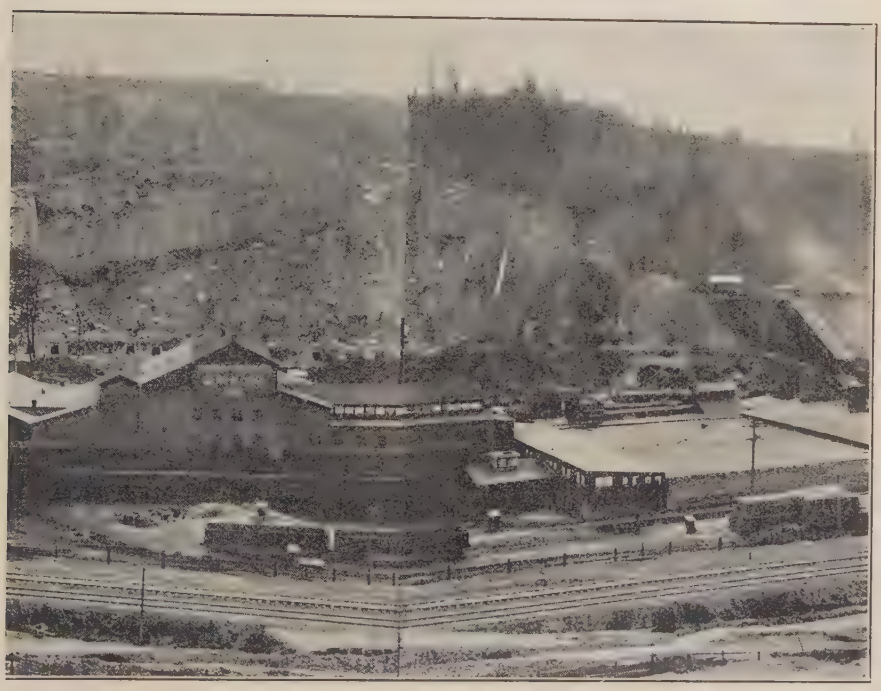
Tunnel Method Now Used

The present method of quarrying and handling the shale is as follows:

The whole face is worked in benches of about 30 ft. Two-inch holes are bored by hand, with a Spry boring machine, about 15 ft. into the

shale, and are loaded with 40 per cent. "Giant" powder and shot seven to ten at a time with a battery. The shale falls on the tunnels shown in Fig. 1 just above the factory roof.

These tunnels—at present about 20 in number—approximate the timbering in a mine crosscut. On 8 by 14 in. mud sills are erected 12 by 14 in. legs, 6 ft. 10 in. apart at the collars and 8 ft. 10 in. apart on mud sills—which are connected by 14 by 16 in. collars, 7 ft. above the mud sills, the whole comprising a set. These sets are spaced 5 ft. apart, and are



PAVING BRICK PLANT AT RENTON.

connected by 3 by 12 in. planks on the sides, which act as lagging, and by 6 by 12 in. bars set 8 in. apart on top of the collars which form the grizzly shown in Fig. 3. Between each set, and underneath this grizzly are constructed two hoppers which together hold a car of shale. These hoppers are closed at the bottom with "needles" similar to those in the bottom of the old-fashioned dump wagons which may be moved easily when the cars are to be filled.

In constructing a tunnel, five sets are set in the solid, and perhaps seven or eight are set up outside. As the shale face of the quarry advances the tunnel is driven to keep five sets ahead of it, the timbers being removed from the outside, and used for driving ahead.

In preparing to shoot on a tunnel, 4 by 12 in. planks are placed crosswise on the grizzly, and these are covered with lumps of shale to a depth of 2 or 3 ft. From 2,000 to 3,000 tons of shale are then shot down on a group of three tunnels, which converts each tunnel, as it were, into a bunker containing from 750 to 1,000 tons of shale. The planks are then removed, a few at a time, from the outermost end of the tunnel and the shale fed through the grizzlies into the hoppers. All lumps are bull-dozed, from one-half to three sticks of dynamite being used, or they are broken up with wedges and hammers; 100-ton lumps are a common occurrence, and the fact that these often fall 125 ft. speaks well for the strength of the construction. The shale between the tunnels, which are placed on 20-ft.

FIG. 2.—REMOVING OVERBURDEN BY HYDRAULICKING.

centers, is left in place until it is time to remove the outer timbers, when it is loaded into cars by hand. This protects the sides of the tunnels and prevents the timbers from buckling—and together with the 2-ft. cushion of shale placed on top of the tunnels before shooting, closely approximates the conditions prevailing in a crosscut in a mine where the "arch" is formed and comparatively weak timbers withstand the weight of millions of tons.

Loading and Conveying

The shale having been shot down covers the tunnels from the quarry face nearly to the outer end of the tunnel. The method of working it down to the grizzly is shown clearly in Fig. 3. The lumps are started with a bar and roll down, some falling through and others having to be

broken by hand. When the hoppers are full, the cars, which are simple boxes 7 by 2½ ft. on the bottom, and flaring to 7 by 3½ ft. on the top, which is 5 ft. above the top of the rail, are filled and run down to the dumping plant on a 3-ft. gauge track. The main lines of the track system are built of 40-lb. rails and all tracks are so graded that the power required to move a full car to the dumping plant is approximately the same as to move an empty back into the tunnel. The trackage and general layout of the tunnels is shown in Fig. 4.

From each direction the main tracks lead into the dumping plant,

FIG. 3.—WORKING SHALE DOWN TO GRIZZLY OVER TUNNEL.

which is equipped with two Link-Belt rotary dumps. These are so arranged that when a trigger is released the whole car rolls over sideways and dumps its contents into a hopper. As soon as the car is empty the center of gravity is thus shifted and the car rolls back into place, too great celerity of action being prevented by a band brake operated by a lever and rope.

The dumping plant is equipped with two hoppers, one for coarse shale equipped with a grizzly and crusher, and the other for fine material.

These hoppers empty the material on a 30-in., 6-ply rubber belt about 50 ft. between pulley centers, which in turn dumps the shale on a similar belt which rises at an angle of 22°, and running at about 300 ft. per minute, delivers the shale to the third floor of the factory.

On reaching the factory, the shale is dumped into a Jeffreys chain conveyor, the flights of which, 24 by 10 in., scrape it along the bottom of a steel-lined box to the openings into the bunkers. There are six of these bunkers, each one of which holds shale sufficient for about 40,000 brick.

Fig. 4.—General Layout of Tracks and Tunnels.

Each bunker feeds a dry pan, or grinding mill, the duty of which is to grind shale sufficient for 30,000 brick in 9 hr.

Each one of the six openings is equipped with a slide, usually left about one-sixth open, which causes all the shale carried along by the chain conveyor to be equally distributed among the six bunkers so that a very even mixture is obtained for each mill. When the bunkers are full, or when it is found desirable to fill the storage, the shale still escapes through one or more of the six openings, but is thrown by means of a hinged door into rope and disk conveyors, 150 ft. long, which run at right angles to the chain conveyor, and carry the shale into the storage shed, the capacity of which is 12,000 tons. This shale is stored in the summer and is mixed with the wet shale in winter in order to make a mixture dry enough to work in the mills efficiently. The stored shale is conveyed to the dry pans by means of underground belt conveyors, whose pits are

covered with plank which may be taken up as the shale is fed to the conveyors.

Efficiency of Tunnel System

Under the hand-loading system, 2-ton capacity, side-dump cars were used, which a crew of three men operated, loading the shale and pushing the cars on an average 100 ft. to the dumping plant, and dumping their own cars. These men worked on contract and averaged 10 cars per man per 10 hr. The output was therefore 2 tons per man per hour.

When the present system was installed the pit was enlarged so that by the time the system was in working order the average haul was 200 ft. The men still worked in crews of three, but each man averaged 1.9, 2.3-ton cars per hour in summer (1.7 in winter), or 4.37 tons per hour per man—more than twice the former output. When the haul reached an average of 300 ft. a mule and driver took over the hauling which increased the tonnage per man per hour to 4.6 tons—enough to pay the cost of the mule and the man. Five men, with the new dumping, conveying and distributing plants handle 700 tons per day as against four men before who handled about 300 tons, so that this cost is just about cut in half by the installation of a modern plant.

The advantages of the new system over the former system may be summed up briefly as follows:

1. The cost of loading, tramming, crushing, conveying and distributing the shale is reduced more than half.

2. A more evenly mixed material is obtained by the system of distribution described and because the shale can be shot down in larger quantities, allowing more choice.

3. The drainage furnished by the tunnels allows the quarry to be worked at times—during the rainy season—when it was formerly impossible, and furnishes a drier and hence more easily ground material. This makes it unnecessary to draw from storage so frequently and so reduces this expense.

Altogether, the system works most successfully. The details have required considerable study and minor changes have been made from time to time, but we believe that, on the whole, this method of quarrying and handling the shale meets the difficulties encountered in working so large and so unusual an undertaking in a manner which is satisfactory.

Discussion

David T. Day, Washington, D. C.—Do you know anything about the character of that clay as to organic matter?

H. Ries, Ithaca, N. Y.—No; it is a dark brownish clay, sandy in its nature. The brownish coloring is probably due to organic matter. Some lignite is also present.

Donald W. Ross.—I have worked at the plant under consideration. I have not seen Mr. Farnham's paper; but in answering the question as to the organic matter, I might say there is practically no organic material scattered through the shale. There are, however, several layers of carbonaceous shale and rather low-grade coal interbedded with the formation.

There are two very serious drawbacks—one which threatens to shut down the plant and another which merely means the expenditure of money. The first is taking care of the sediment from the washing away of the overburden. The government is doing extensive work at different points about Lake Washington with a view to creating a fresh-water harbor for Seattle. I understand that they keep at the Denny-Renton people to keep the channel of Cedar river clear. The Denny-Renton Co. can hardly help filling up the channel so long as they continue their hydraulic work, so this is probably one of their most serious problems. The other is this: There is a large percentage of what I should call micaceous shale or sandstone—a sandstone which contains a large amount of light-colored mica. Of course only a limited amount of this can be used in the manufacture of paving brick. As they shoot down the bank after the overburden of gravel and loam is washed away, these large boulders of micaceous sandstone come down. The strata have apparently been crushed where the clay pit is located so that the exact formation cannot be readily determined. These pieces of sandstone, 5 or 6 ft. in diameter, have to be blasted and removed from the shale. They bother both in the open pit and in the tunnel mining. In the tunnel mining of the shale blocks frequently cover the holes in the roof of the tunnel through which the shale is loaded into cars.

David T. Day.—About how many tons of that bituminous shale do they get in a day?

Donald W. Ross.—I believe, owing to a disagreement with the miners, that they have dug none of this coal during the last three years. The coal is of rather low grade, compared with other coals of the region, and is taken from a vein some 3 or 4 ft. thick, which dips approximately 15° to 30° to the southeast. The other coal vein is controlled by the Seattle Electric Co., and is approximately parallel to this vein and about 600 ft. below it. Dr. Fettke tells me that it approximates 15 per cent. ash. I have seen similar coals from other mines that contained as high as 60 per cent. ash.

H. Ries.—That coal you speak of is not in the main bank?

Donald W. Ross.—The entrance to this Denny-Renton coal mine is situated about 300 ft. north of the shale pit toward the town of Renton. I am inclined to believe that the coal vein, due to its dip, passes beneath the floor level of the pit.

Barytes as a Paint Pigment

BY H. A. GARDNER, AND G. B. HECKEL, WASHINGTON, D. C.

(Pittsburgh Meeting, October, 1914)

THE principal use of barium sulphate is as an inert paint pigment. For this purpose, the ground material is used both in its natural and in its artificial forms. Probably the largest amount is used in the production of a chemically precipitated pigment called lithopone which consists of 70 parts of barium sulphate and 30 parts of zinc sulphide. An enormous tonnage of this pigment is at the present time being used in the manufacture of interior flat wall paints of the oil type, such paints having almost entirely supplanted the use of corroded white lead, which was at one time used for interior painting. The more dense nature, lower price and sanitary value of lithopone were responsible for this change. Lithopone is, moreover, used to a large extent in the manufacture of oil cloth, shade cloth, and linoleum. Its whiteness makes it of particular value for this purpose. Barium sulphate is also used of itself in admixture with other pigments in the manufacture of prepared paints. The chemically precipitated form of barium sulphate, which is generally called *blanc fixe*, is also used for this purpose.

Much controversy has raged in the paint world over the question of inert pigments such as barytes, silica, asbestine, and china clay. These pigments are probably called inert because they have no chemical action upon the linseed oil in which they are ground to produce paints, and on account of the fact that they become transparent in oil. It is well known that in the manufacture of paints, opaque white pigments, such as basic carbonate white lead, basic sulphate white lead, zinc oxide, etc., are generally used as the grinding base. These base pigments, when ground in oil, react therewith to some slight extent, and they produce white paints which have excellent hiding or obscuring value. The inert pigments so-called, when ground in oil and spread out into a thin film, form transparent coatings which have no hiding power. It has been contended, therefore, that the use of barytes or similar inert pigments in paints is resorted to simply for the purpose of cheapening the cost of the product. It is, of course, true that the natural tendency of manufacturers in the past has been to use inert pigments liberally, on account of their low cost, and the consumer has in consequence frowned upon this practice and has been prone in some instances to condemn the use of paints which contain

even small amounts of inert pigments. That such inert pigments do have a proper and legitimate place in the manufacture of prepared paints, has been shown by practical tests. The degree to which they may be used, however, is a question which some believe not to have been fully determined. Several years ago, at the laboratories of the Pennsylvania Railroad Co., in Altoona, Pa., Dr. Charles B. Dudley conducted a long series of carefully made paint tests on a practical scale. From the results of these tests, Dr. Dudley formulated a series of fundamental principles governing the quantity and use of inert pigments in prepared paints. Briefly paraphrased, these principles may be stated as follows:

> In a white paint, the desired color having been provided for by the use of opaque pigments, there may then be added to the paint as much inert pigment as can be used without destroying the practical opacity of the paint. In a white paint made of lead and zinc pigments, the amount of inert pigment to be added may probably run as high as 20 per cent., but a more dense paint will be obtained by the use of not over 10 to 15 per cent. In a colored paint, such as an iron oxide red, there may be used as high as 50 per cent. of inert pigments, without detracting from the color of the paint. In a colored paint such as a chrome green, there might be used as high as 80 per cent. of an inert pigment.

The use of barytes in the manufacture of chrome green conspicuously illustrates the function of an inert pigment. Chemically pure green is prepared by simultaneous precipitation of lead chromate and Prussian blue. This pigment is used as a tinting color, and when it is used alone it is apt to produce a dense, dingy green. In order to make a more satisfactory green, the color is precipitated upon a base of barium sulphate in the proportion of 20 parts of color to 80 parts of barium sulphate. Such a color is brilliant and clear, producing bright shades when added to a white base. This statement also holds true with other colors, especially the adjective dyes precipitated on a barium sulphate base. The para-nitraniline reds and other permanent vermilions may be cited as examples of brilliant colors which are now precipitated in the course of manufacture upon a base of barium sulphate or similar inert pigment. The legitimate office, therefore, of barytes as an inert pigment in the manufacture of colors is to diffuse the color by its presence and brighten and make more clear the tints which are prepared from it as a base.

It is a fact, which at first glance appears paradoxical, that the incorporation of small amounts of these transparent, colorless, inert pigments in a paint containing very finely divided opaque pigments, such as zinc oxide, lampblack, etc., increases the opacity of the paint coatings made from them. The reasons, however, are obvious. In the first place, the proportion of liquid in the vehicle of the film is reduced, on account of the fact that barytes grinds to a workable base with a very small amount of linseed oil, whereas many other pigments used in the manufacture of paints require a more substantial quantity of oil. It is

the amount of oil in a paint which materially affects its hiding power. Moreover, the thickness of a paint coating is to a great extent increased when it contains an inert pigment such as bartyes, and the thicker the coat the more dense will be the paint.

The chemical effect of inert pigment is also a matter of great importance to the painter. This effect varies with the structure of the pigment. In the case of pigments such as bartyes, silica, etc., the effect is to impart "tooth" under the brush. "Tooth" is an expression used to describe the brushing action of a paint, which indicates that the paint is being taken up by the wood or other surface to which it is applied. Paints made solely from finely divided pigments such as zinc oxide or lampblack would be deficient in tooth.

From the foregoing, it is apparent that there should be no technical objection to the use of bartyes in paints. The only question is an economic one, and involves two considerations: The first is that barytes and other inert pigments shall not be sold except under their true names and shall not be used as adulterants, using that term in the sense given it by the American Society for Testing Materials, as follows: "Adulterant—A substance partially substituted for another." The other consideration is that the amount of bartyes shall be such that it will not impair the opacity of the paint under the conditions of application, which implies that three coats shall effectively hide the underlying surface to which it is applied.

Data on Barytes

Occurrence: As rock barite and as the gangue in zinc and lead ores.

Principal sources: Missouri, Kentucky, Virginia, and Tennessee.

Structure: Natural, crystalline; artificial, amorphous.

Color: White, transparent.

Usual method of production: "Gophering."

Principal use: As a pigment.

Preparation of pigment: Breaking, comminution, washing with acid to remove iron, washing, grinding, water floating or sedimentation.

Specific gravity of pigment: 4.45.

Formula: $BaSO_4$.

Desirable qualities of barytes as a paint pigment: Softness, whiteness, fineness.

INDEX

[NOTE.—In this Index the names of authors of papers are printed in small capitals and the titles of papers in italics. Casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not in this volume, are followed by bracketed page-numbers only.]

Abel, Frederick: investigations of coal-dust explosions, [553].
Accidents (fatal):
in metal-mining States (1911–12), 765.
in Michigan iron mines (1901–1911), 765.
Adsorption phenomena in fractionation of oils, 844, 849, 853.
Aerial Tramway for Mining Cliff Coal (GIBSON), 679–688.
Afterdamp: composition, 565.
AIKEN, W. A.: *Discussion on Finishing Temperatures and Properties of Rails*, 321.
Air:
decarbonization of steel by annealing in, 408.
heat of compression, 101.
weight at various temperatures, 102.
in mines: contamination by exhaust of gasoline locomotives, 771.
Air compressors:
balancing end thrust, 113.
constant volume governor for, 112, 136.
turbo driven, 104, 137.
Alabama: iron-ore resources, 200.
Alkali content of blast-furnace gas, 54.
Allotropic theory:
electrolytic iron, 515–531.
manganese steel, 484, 497, 500, 501–514.
Alloys: iron-nickel: magnetic qualities, 480.
Aluminum: experiments with, in deoxidation of cast iron, 353.
American Institute of Mining Engineers: Pittsburgh meeting, October, 1914, vii.
American Steel-Rail Situation (HUNT), 340–343.
Analyses:
afterdamp, 566.
blast-furnace flue dust, Duquesne furnaces, 8–10, 14.
blast-furnace gas, 5, 6, 92.
coal: Fu Chuan district, China, 696.
iron ore: Brazil, 143, 150, 151.
Hong-Kong, China, 243.
Lake Superior, 226.
Lux mass for gas filters, 66.
pig steel, 176.
electric steel furnace slag, 176.
salts in water of Great Salt Lake, 938.
steel castings, annealed and heat treated, 425–430.

Analytic methods for sulphur in pig iron, 349.
Annealing:
effect of chemical nature of packing material and method of packing, 421.
effect of repeated, in decarbonization of tool steel, 411
inefficiency of commercial, 425, 432.
manganese steel, 489.
tool steel: in air, 408
in carbon dioxide, 407.
in carbon monoxide, 410.
in lead oxide, 412.
in oxygen, 410
in steam, 411.
Appraisal of Coal Lands for Taxation (CHANCE), 625–630; *Discussion* (EVANS), 639; (GRIFFITH), 637, 639; (HALL), 638; (NORRIS), 631–636; (TAYLOR), 637–639.
Argon in natural gas, [835].
Arnou, M. G.: direct reduction of iron ore in the electric furnace, [169].
Asbestic mortar, 962.
Asbestos:
mineralogy, 965.
modes of occurrence, 966.
production and value (1910), 961.
Georgia: associated rocks, 967.
mining and preparation, 971.
types found, 967.
uses, 972.
New York: occurrence at Tompkinsville, 963.
Quebec: mining and dressing, 958.
vein structure and origin, 956.
uses, 959.
Asbestos Deposits of Georgia (HOPKINS), 964–973.
Asbestos in Southern Quebec (DRESSER), 954–961; *Discussion* (DAY), 962; (DRESSER), 961–963; (LINDGREN), 962; (RIES), 962; (SPILSBURY), 961; (WHEELER), 963.
Asbestos Mining & Manufacturing Co.: asbestos mine, Hollywood, Ga., 970.
Asphalt deposits:
Mexico: Cerro Azul, 866.
Trinidad, W. I.: Pitch Lake, 865.
Venezuela: Bermudez Lake, 865.
ASTON, JAMES: *Discussion on Finishing Temperatures and Properties of Rails*, 317–319.
Atkinson W. N. and J. B.: coal-dust explosions, 554.
AUSTIN, L. S.: *Discussion on The Need of Uniform Methods of Sampling Lake Superior Iron Ore*, 215–230.
Austria-Hungary: magnesite deposits, 892.

Bakker, G.: capillary pressures, [851].
BANNER, OTTO: *Discussion on Turbo Blowers for Blast-Furnace Blowing*, 139.
BARR, JAMES A.: *Tennessee Phosphate Practice*, 917-933.
Barytes as a Paint Pigment (GARDNER and HECKEL), 983–985.
BELAIEW, N.: *Discussion on Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron*, 527.
Belgium: coal-dust explosions (1887–1904), 615.
BENEDICKS, CARL: *Discussion on Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron*, 526.

Benzol: recovery from coke-oven gas, 66.
Bethlehem Steel Co., South Bethlehem, Pa.: duplex steel plant, 253.
Bibliography: manganese steel, 494.
Birkinbine, John: *The Reserves of Iron Ore for the United States*, 197–203.
Black Hawk Coal Co., Carbon County, Utah: mine and mining methods, 675.
Blast-furnace gas: see Gases.
Blast-furnace practice:
advantages of steady blast, 110, 127, 131, 134.
apparatus for measuring variations in blast pressure, 107.
blast requirements, 93.
conditions producing good and bad irons, 372.
constant volume governor for blast, 112, 136.
heat balances of coke and charcoal furnaces, 369.
influence of oxygen, nitrogen, etc., on cast iron, 344–404.
operating as gas producer, [57], [87], [88].
operating with a turbo blower, 90–142.
sampling iron ore, 207.
variation of blast pressure with turbo and reciprocating blowing engines, 109.
Blast furnaces: Brazil, 146.
Blasting fuse: Cordeau Bickford, 738–754.
Boilers:
efficiency ratings of coal and gas fired, 118.
tests with blast-furnace gas, Duquesne furnaces, 37.
Book Cliffs Coal Field, Utah (Lewis), 658–678.
Book Cliffs coal field:
aerial tramway, 679–688.
map, 662.
Braman, H. S.: *Discussion on Turbo Blowers for Blast-Furnace Blowing*, 140.
Brazil:
blast furnaces, 146.
coal deposits, 157.
iron-ore deposits, 143–160, 201.
maps: general, 144.
Minas Geraes: iron-ore deposits, 148.
railroads, 155.
Breth, Norman V.: *Discussion on Shot Firing in Coal Mines by Electric Circuit from the Surface*, 731–733.
Brick (paving): manufacture from shale at Renton, Wash., 974–982.
Brinell hardness of manganese steel, 484.
Bureau of Mines (U. S.): experimental mine, Bruceton, Pa., 559.
investigations of coal-dust explosions, 552–587.
safety work in the Lake Superior iron region, 762.
Burgess, George K.: *Discussions: on Manganese Steel and the Allotropic Theory*, 511–512.
on Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, 520.
Burgess, George K., Crowe, J. J., Rawdon, H. S., and Waltenberg, R. G.: *Finishing Temperatures and Properties of Rails*, 302–306, 322–326.
Burnham, M. H.: mine valuation, [188], [191].
Burrell, G. A.: *Discussions: on The Capillary Concentration of Gas and Oil*, 848, 850.
on Gas and Oil Wells through Coal Seams, 880.
Bushong, F. W.: *Discussions: on The Capillary Concentration of Gas and Oil*, 850.
on Refining Petroleum by Liquefied Sulphur Dioxide, 827.

Cady and McFarland: helium and argon content of natural gas, [835], [856].
California:
magnesite deposits, 892, 898.
salt industry, 940.
California Salt Co., Alvarado, Cal.: manufacture of salt from sea water, 940.
Campbell, William: *Discussion on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 392.
Canada asbestos deposits, 954–964.
Capillary Concentration of Gas and Oil (Washburne), 829–842; *Discussion* (Burrell), 848, 850; (Bushong), 850; (Chance) 846; (Day), 843–846; (Johnson), 842, 847; (Washburne), 850–858; (Wheeler), 847.
Capillary openings: maximum and minimum diameters, 833, 851.
Carbon:
absorption by iron, 372, 393.
absorption by steel: effect of variations in coke, limestone, and fluorspar in charge, 167.
affinity for oxygen increases with increase of temperature, [367].
effect in deoxidation of iron, 372.
elimination from tool steel, 405–423.
Carbon dioxide:
decarbonization of steel in, 407.
decomposition of zinc by, 421.
in mine air: effect on health of miners, 771.
Carbon monoxide:
decarbonization of steel by annealing in, 410.
in blast-furnace gas: proportion to carbon dioxide, 4.
in mine air: proportion injurious to health of miners, 771.
Carnegie Steel Co.: gas-cleaning plants, Duquesne and Braddock, Pa., 3–55.
Carpenter, H. C. H.: *Discussions: on Notes on the Plastic Deformation of Steel During Overstrain*, 547–548.
on Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, 517–519.
Carpenter and Edwards: hardening of metals, [522].
Carpenter's electrolytic iron: heating and cooling curves, 515–531.
Cast iron: see Iron.
Castings (steel): heat treatment, 424–436.
Castle Gate coal mine, Carbon County, Utah, 661.
Castle Valley Coal Co., Mohrland, Utah: mine and mining methods, 673.
Catani, R.: production of steel from ore in the electric furnace, [161].
Chance, Edwin M.: *Discussion on The Capillary Concentration of Gas and Oil*, 846.
Chance, H. M.: *The Appraisal of Coal Lands for Taxation*, 625–630.
Channing, J. Parke: mine valuation, [190].
Chapin iron mine Iron Mountain, Mich.: centrifugal motor-driven pumps, 779–808.
Chart for computing interest (6 per cent., 1 to 40 years), 233.
Child, Edward T.: *Discussion on Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes*, 89.
Chili: iron-ore resources, [201].
China:
iron-ore deposit, Hong-Kong New Territory, 236–245.
map, 237.
tin and coal deposits, Fu Chuan district, 689–697.
Chlorides in Oil-Field Waters (*Trans.*, xlviii, 687 to 694); *Discussion* (Washburne), 883–889.

Chrome-nickel steel: effect of treatment on magnetic qualities, 484.
Chromium: effect on quality of cast iron, 380.
Cirkel, F.: asbestos mining and preparation, [958].
Clark and Hall: investigation of coal-dust explosions, [552].
Clark, H. H., and Rice, George S.: *Shot Firing in Coal Mines by Electric Circuit from the Surface*, 723–731, 733, 736–737.
Clarke: phosphorus content of lithosphere, [905].
Cleveland Chemists' standard method of sampling iron ore, 205.
Coal deposits:
  Brazil, 157.
  China: Fu Chuan district, 689–697.
  Pennsylvania: Panther Creek Valley, 698.
    Pittsburgh field, 640–657.
  Utah: Book Cliffs field, 658–678.
  Washington: Denny-Renton mine, [982].
Coal-Dust Explosion Investigations (Taffanel), 588–593.
Coal-dust explosion:
  investigations of, 552–624.
  preventive methods, 567.
Coal lands: appraisal for taxation, 625–639.
Coal-Mine Explosions Caused by Gas or Dust (Eavenson), 594–618; *Discussion* (Eavenson), 621, 623; (Grady), 622; (Griffith), 621; (Hall), 621; (Norris), 622, 624; (Peters), 624; (Rice), 619, 624; (Schellenberg), 622; (Stoek), 621; (Taylor), 623; (Wilson), 622; (Young), 620.
Coal mining: see Mining methods.
Coke:
  effect of variation in charge on carbon absorption in steel, 167.
  quantity required for iron production by various districts, 648.
Coke-oven gas: see Gases.
Coke ovens:
  Connellsville field, Pa., 643.
  Frick Coke Co., 644.
Coke vs. charcoal for electric smelting of iron ore, 185.
Cone, Edwin F.: *Discussion on The Heat Treatment of Steel Castings*, 432.
Connellsville coal field: unmined acreage and number of coke ovens, 643.
Conservation of Lake Superior iron ore, 231–235.
Consolidated Fuel Co., Hiawatha, Utah: mines and mining methods, 676.
Converter-bottom drying ovens, Bethlehem Steel Co., 261.
Converters (steel):
  Bethlehem Steel Co., 260.
  Dominion Iron & Steel Co., 252.
  Pennsylvania Steel Co., 264.
  Tennessee Coal, Iron & Railroad Co., 250.
Cordeau Bickford (detonating fuse), 738–754.
Cornwall Ore Bank Co., Cornwall, Pa.:
  blasting record sheet, 740.
  experience with Cordeau Bickford (detonating fuse), 738–754.
Corrosion of pumps by acid water: cement lining to prevent, 806.
Costs:
  blast-furnace, coke-oven, and producer gas, 73.
  cleaning blast-furnace gas, 41.
  coal mining: variation for beds of different thicknesses, 633.
    variation for different rates of output, 635.

Costs:—Continued.
Pittsburgh field, Pa., 652.
electric power from coke-oven and blast-furnace gas, 59, 76–83.
mining phosphate rock, Tennessee, 920, 930.
refining petroleum (Edeleanu process), 824.
Crawford, J.: electric smelting at Heroult, Cal., [170].
Crawford, John, Jr.: *Discussion on Pig Steel from Ore in the Electric Furnace*, 180–187.
Critical ranges of manganese steel, 437–475.
Crowe, J. J., Rawdon, H. S., Waltenberg, R. G., and Burgess, George K.: *Finishing Temperatures and Properties of Rails*, 302–306, 322–326.
Cuba: iron-ore resources, [201].
Cutler, F. G.: volumetric efficiency of blowing tubs, [141].

Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces (Diehl), 3–46; *Discussion* (Diehl), 55; (Grammer), 54; (Maccoun), 46–54; (Richards), 55.
Day, David T.: *Discussions: on Asbestos in Southern Quebec*, 962.
on The Capillary Concentration of Gas and Oil, 843–846, 849.
on Gas and Oil Wells through Coal Seams, 878, 880, 881.
on The Occurrence, Preparation and Use of Magnesite, 897–900.
on The Oil Fields of Mexico, 863.
on Quarrying Shale by the Tunnel System, 981, 982.
on Salt Making by Solar Evaporation, 951, 952, 953.
Decarbonization:
of cast iron: effect of chemical nature of packing material and method of packing, 421.
of manganese steel, 496, 499.
of tool steel, 405–423.
Denny-Renton Clay & Coal Co., Renton Wash.: shale quarry and paving-brick plant, 974–982.
Deoxidation of cast iron: effect on quality, 353, 387.
Derby, A. O.: iron-ore deposits, Sao Paulo, Brazil, [145].
Detonating fuse (Cordeau Bickford), 738–754.
Diamondiferous peridotite, Arkansas, [884].
Diehl, A. N.: Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces, 3–46, 55.
Diffusion: fractionation of oils by, 844, 849, 853.
Diller, J. S.: modes of occurrence of asbestos, 966.
Dominion Iron & Steel Co., Nova Scotia: duplex steel plant, 252.
Dresser, John A.: *Asbestos in Southern Quebec*, 954–963.
Drying ovens, Bethlehem Steel Co., 261.
Dudley, C. B.: inert pigments in prepared paint, 984.
Dudley, P. H.: *Discussion on Finishing Temperatures and Properties of Rails*, 310–314.
Duplex process of making steel: plants for, 246–269.
Dust explosions: investigations, 552–624.
Dynamite: rates of detonating, 750.

Eavenson, Howard N.: *Coal-Mine Explosions Caused by Gas or Dust*, 594–618, 621, 623.
Edeleanu, L.: *Refining Petroleum by Liquefied Sulphur Dioxide*, 809–827.
Edgar Thomson Steel Works, Braddock, Pa.: gas-cleaning plant, 46.
Edwards and Carpenter: hardening of metals, [522].
Ehrhardt & Sehmer, Saarbruecken, Germany: large coke-oven gas engines, 63.
Electric-furnace production of pig steel, 161–187.
Electric power from blast-furnace and coke-oven gas: cost of production, 59, 76–83.

Electric power stations at steel plants: gas engine vs. steam turbine for driving generators, 121, 126.
Electric smelting of iron ore, 161–187.
Electrical shot firing from the surface in coal mines, 723–737.
Electrically driven centrifugal pumps at Chapin iron mine, Iron Mountain, Mich.: tests, 779–808.
Electrolytic iron: heating and cooling curves, 515–531.
Emmons, J. V.: *The Surface Decarbonization of Tool Steel*, 405–420, 423.
Engines:
blowing: comparison of efficiency of steam turbine and steam and gas reciprocating engines, 90–103, 116, 129.
method of rating, 105, 131.
turbine, 90–142.
gas: coke-oven vs. blast-furnace gas fuel, 56–89.
for blast-furnace blowing, 119, 121.
large German units, 63, 84.
remedies for corrosion of cylinders by sulphuric acid, 65.
tests with blast-furnace gas, Duquesne furnaces, 38.
Ennis, A. J.: *Discussion on The Need of Uniform Methods of Sampling Lake Superior Iron Ore*, 214.
Evans, Mr.: *Discussion on The Appraisal of Coal Lands for Taxation*, 639.
Explosions:
coal-dust and gas: investigations of, 552–624.
coal mine: preventive methods, 567.
Explosives: rates of detonation, 750.

Farnham, Dwight T.: *Quarrying Shale by the Tunnel System*, 974–981.
Ferguson, James C. H.: *Rolled Steel Roll Shells*, 290–301.
Ferric oxide: temperature of decomposition, 422.
Fertilizers:
acid phosphate: objectionable impurities, 903, 920.
preparation for market, 914, 931.
Filtration: fractionation of oils by, 844, 849, 853.
Finishing Temperatures and Properties of Rails, (Burgess, Crowe, Rawdon, and Waltenberg), 302–306; *Discussion* (Aiken), 321; (Aston), 317–319; (Burgess), 322–326; (Dudley), 310–314; (Gibbs), 320; (Kenney), 319; (Sauveur), 306; (Waterhouse), 314–316; (Webster), 307–310; (Wickhorst), 316.
Firedamp and dust explosions in coal mines, 594–624.
First-aid and rescue work in the Lake Superior iron region, 760.
Florida: phosphate deposits, 907.
Flue dust:
blast furnace: analyses, 8, 9, 10, 14.
chemical and physical characteristics, 8.
quantity in blast-furnace gas, 12.
settling tests, 10.
Fluorspar: effect of variation in charge on carbon absorption in steel, 169.
Fohl, W. E.: *Discussion on Gas and Oil Wells through Coal Seams*, 870–873, 881.
France, coal-mine explosions (1814–1904), 614.
Freyn: Heinrich J.: *Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes*, 56–84, 87–89.
Frick Coal Co.:
coal-mining methods, Pittsburgh field, 654.
coke ovens in Connellsville field, Pa., 644.

Friedrich-Wilhelms-Huette, Muelheim-Ruhr, Germany: large coke-oven gas engines, Prinz Regent colliery, 63.
Fritsch, J.: manufacture of chemical manures, [903].
Furnaces: see Blast furnaces.
Furst, J. K.: *The Plant of the Duplex Process for Making Steel*, 246–267.
Fuse for blasting: Cordeau Bickford, 738–754.

Galloway, William: investigation of coal-dust explosions, [552].
Gardner, H. A., and Heckel, G. B.: *Barytes as a Paint Pigment*, 983–985.
Garforth, William E.: investigation of coal-dust explosions, 555.
Gas and oil: capillary concentration, 829–858.
Gas and Oil Wells through Coal Seams (A Discussion): (Burrell), 880; (Day), 878, 880, 881; (Fohl), 870–873, 881; (Hall), 876–878, 879; (Hood), 879; (Layton), 873–875, 876, 881. (Rice), 870, 875–876.
Gas-cleaning apparatus:
  Diehl spray type, 18, 49.
  fans, 21.
  Murray cleaner, 46.
  settling basins, 23.
  scrubbers, 18.
  Theisen washers, 22.
Gas cleaning at Duquesne furnaces and Edgar Thomson Steel Works, 3–55.
Gas engines: see Engines.
Gas explosions in coal mines, 594–624.
Gas fields: rock pressures, 847, 854.
Gas fuel: metallurgical uses, [69].
Gases:
  afterdamp: composition, 565.
  blast furnace: advantages of mixture with coke-oven gas for use in engines, 68.
    available heat, 6.
    clean vs. dirty, 36, 52.
    cleaning, 3–55.
    composition, 4, 71, 92.
    cost of cleaning, 41.
    cost of producing, 73.
    dust content, 12, 34.
    moisture content, 5, 34, 50.
    sampling for analysis, 31.
    standard form for tests, 33.
    utilization for power purposes, 56–89, 121.
  carbon monoxide and dioxide in exhaust of gasoline locomotives, 773.
  coke-oven: advantages of mixture with blast-furnace gas for use in engines, 68.
    elimination of sulphur, 65.
    Lux mass for eliminating sulphur, 66.
  producer: composition, 72.
    cost of producing, 73.
    utilization for power purposes, 56–89.
  natural: argon and helium content, [835], [856].
    solubility in oils, 846, 848.
Gasoline Locomotives in Relation to the Health of Miners (Hood), 771–775; *Discussion* (Hood), 777; (Norris), 775; (Rice), 775; (Taylor), 776; (Whildin), 776.

Geology:
China: Hong-Kong and vicinity, 236–245.
Kwang Si province, 689.
Florida: phosphate deposits, 907.
Georgia: asbestos deposits, 966.
Mexico: oil fields, 860.
Pennsylvania: Panther Creek Valley, 698.
Quebec: asbestos deposits, 955.
Tennessee: phosphate deposits, 917.
Utah: Book Cliffs coal field, 658.
Georgia: asbestos deposits, 964–973.
Gibbs, A. W.: *Discussion on Finishing Temperatures and Properties of Rails*, 320.
Gibson, Arthur E.: *An Aerial Tramway for Mining Cliff Coal*, 679–688.
Gilbert, G. K.: analyses of water of Great Salt Lake, 938.
Goerens, P.: properties of strained steel, [493].
Grady, W. H.: *Discussion on Coal-Mine Explosions Caused by Gas or Dust*, 622.
Grammer, F. Louis: *Discussions: on Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces*, 54.
on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements, 393–396.
on Notes on Blast-Furnace Operation with a Turbo Blower, 102.
Graphite: distribution in cast iron and effect on quality, 357, 387, 392, 397.
Great Britain: coal-mine explosions (1803–1912), 607.
Great Salt Lake:
composition of salts, 938.
production of salt from, 936.
Greece: magnesite deposits, island of Euboea, 890.
Grey, L. C.: economic possibilities of conservation, [231].
Griffith, William: *Discussions: on The Appraisal of Coal Lands for Taxation*, 637, 639.
on Coal-Mine Explosions Caused by Gas or Dust, 621.
on Investigations of Coal-Dust Explosions, 585.

Hadfield, Sir Robert: *Manganese-Steel Rails*, 327–333, 335–339.
Sound Ingots, 270–289.
Discussions: on Manganese Steel, with Especial Reference to the Relation of Physical Properties to Microstructure and Critical Ranges, 471–475.
on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, 521–526.
Hadfield, Sir Robert, and Hopkinson, B.: *Research with Regard to the Non-magnetic and Magnetic Conditions of Manganese Steel*, 476–500.
Hall, Clarence: *Discussion on A New Safety Detonating Fuse*, 747.
Hall, Henry: coal-dust explosions, [555].
Hall, John H.: *Discussions: on The Heat Treatment of Steel Castings*, 433–435.
on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements, 387–389.
Hall, R. D.: *Discussions: on The Appraisal of Coal Lands for Taxation*, 638.
on Coal-Mine Explosions Caused by Gas or Dust, 621.
on Gas and Oil Wells through Coal Seams, 876–879.
Hall and Clark: investigation of coal-dust explosions, [552].
Harder, E. C.: *The Iron Industry in Brazil*, 143–156, 159–160.
Hatfield, W. H.: decarburization of iron-carbon alloys, [405].
Hayes and Ulrich: origin of Tennessee phosphate deposits, [911].

Heat Treatment of Steel Castings (Young, Pease, and Strand), 424–430; *Discussion* (Cone), 432; (Hall), 433–435; (Pease), 435; (Strand), 431.
Heckel, G. B., and Gardner, H. A.: *Barytes as a Paint Pigment*, 983–985.
Helium in natural gas, [835], [856].
Henry, Philip W.: *Discussion on The Oil Fields of Mexico*, 864–867.
Hess, R. A.: economic aspects of conservation, [231].
Hibbard, Henry D.: *Discussions: on The Influence on the Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 385, 393.
on Notes on the Plastic Deformation of Steel During Overstrain, 548.
on The Plant of the Duplex Process for Making Steel, 268.
Higgins, Edwin: *The Safety Movement in the Lake Superior Iron Region*, 755–770.
Hofman, H. O.: *Discussion on The Surface Decarbonization of Tool Steel*, 422.
Honda, K.: transformations of iron and steel, [521].
Honda and Takagi: transformation points of iron and steel, [493].
Hong-Kong, China: iron-ore deposits, 236–245.
Hood, O. P.: *Gasoline Locomotives in Relation to the Health of Miners*, 771–775, 777.
Discussions: on Gas and Oil Wells through Coal Seams, 879.
on A Test of Centrifugal Motor-Driven Pumps, 803.
Hook, J. S.: geology of Tennessee phosphate deposits, [917].
Hopkins, Oliver B.: *Asbestos Deposits of Georgia*, 964–973.
Hopkinson, B., and Hadfield, Sir Robert: *Research with Regard to the Non-magnetic and Magnetic Conditions of Manganese Steel*, 476–500.
Hot-blast stoves: efficiency of, 52.
Howe, Henry M.: *Discussions: on the Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 385.
on Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, 519.
on Research with Regard to the Non-magnetic and Magnetic Conditions of Manganese Steel, 497.
Howe, Henry M., and Levy, Arthur G.: Notes on the *Plastic Deformation of Steel During Overstrain*, 532–549.
Howell, S. P.: *Discussion on A New Safety Detonating Fuse*, 748–751.
Humidity as a preventive of coal-dust explosions, 567.
Hunt, Robert W.: *The American Steel-Rail Situation*, 340–343.
Hydrocarbon content of blast-furnace gas, 5.
Hydrogen:
effect in decarbonization of steel, 406.
variation in content of blast-furnace gas, 5.

Independent Coal & Coke Co., Kenilworth, Utah: mine and mining methods, 670.
Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements (Johnson), 344–383; *Discussion* (Campbell), 392; (Grammer), 393–396; (Hall), 387–389; (Hibbard), 385, 393; (Howe), 385; (Johnson), 396–404; (Moldenke), 383–385; (Sauveur), 389–391; (Strand), 387; (Stoughton), 391; (Sweetser), 392; (Webster), 387.
Ingot molds: for steel roll shells, 293.
Ingots (steel): Hadfield method of manufacture, 330.
Inland Crystal Salt Co., Saltair, Utah: manufacture of salt from water of Great Salt Lake, 936.
Interest: chart for computing, 233.
Investigation of Coal-Dust Explosions (Rice), 552–585; *Discussion* (Griffith), 585; (Rice), 585–587.
Ipanema, Brazil, iron-ore deposits, 145.
Iron and nickel oxysulphides: penetration of iron by, 349.

Iron:
cast: absorption of carbon, 372, 393.
decarbonization by iron oxide, 407, 422.
deoxidation with aluminum and titanium: experiments, 353.
effect of carbon in deoxidation, 372.
high vs. low carbon, 376.
influence of oxygen, nitrogen, and other elements on quality, 344–404.
electrolytic: allotropic theory, 515–531.
heating and cooling curves, 515–531.
malleable: analogy with high-oxygen iron, 358.
effect of nature of packing material and method of packing, 421.
pig: rate of increase of production, 646.
Iron Industry in Brazil (Harder), 143–156; *Discussion* (Harder), 159–160; (White), 157–159.
Iron mines: valuation of, 188–196.
Iron-nickel alloys: magnetic qualities, 480.
Iron ore:
direct production of steel from, in the electric furnace, 161–187.
Lake Superior: conservation, 231–235.
need of uniform methods of sampling, 204–230.
United States reserves, 197–203.
Iron-ore deposits:
Brazil, 143–160.
China: Hong-Kong New Territory, 236–245.
Iron oxide:
decarbonization of cast iron by, 407, 422.
decomposition by heating in vacuum, 422.
temperature of decomposition, 422
Itabirite iron ores, Brazil, 151.
Iversen, L.: *Discussion on Turbo Blowers for Blast-Furnace Blowing*, 128–131.

Johnson, C. M.:
decarbonization of steel by rust, [405].
decomposition of ferric oxide, [423].
Johnson, J. E., Jr.: *The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 344–383, 396–404.
Discussions: on Notes on Blast-Furnace Operation with a Turbo Blower, 96, 98, 100, 101.
on Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes, 87, 88.
on Turbo Blowers for Blast-Furnace Blowing, 131–134, 140.
Johnson, Roswell H.: *Discussion on The Capillary Concentration of Gas and Oil*, 842, 847.
Johnson, W. McA.: *Discussions: on The Plant of the Duplex Process for Making Steel*, 267.
on Turbo Blowers for Blast-Furnace Blowing, 128.
Johnston and Adams:
capillary movement of water, [836].
capillary pressures, [834], [852].

Keeney, Robert M.: *Pig Steel from Ore in the Electric Furnace*, 161–179.
Kelly, William: *Discussions: on The Need of Uniform Methods of Sampling Lake Superior Iron Ore*, 210–211.
on A Test of Centrifugal Motor-Driven Pumps, 802, 803, 804, 805, 807.

Kenney, E. F.: *Discussion on Finishing Temperatures and Properties of Rails*, 319.
Kohlmeyer, E. G.: temperature of decomposition of ferric oxide, 422.
Kuhn, H. A.: *The Pittsburgh Coal Field in Western Pennsylvania*, 640–657.

Lake Superior iron ore:
conservation, 231–235.
need of uniform methods of sampling, 204–230.
Lake Superior iron region: safety movement, 755–770.
Lake Superior Mining Institute: safety work, 762.
Lamb, Richard: *Discussion on Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes*, 84–87.
Lane, Alfred C.: *Discussion on Notes on Conservation of Lake Superior Iron Ores*, 235.
Langer, Paul: mechanical features of large gas engines, [64].
Layton, M. B.: *Discussion on Gas and Oil Wells through Coal Seams*, 873–875, 876, 881.
Lead oxide: decarbonization of steel by annealing in, 412.
Le Chatelier, Henry: *Discussions: on Notes on the Plastic Deformation of Steel During Overstrain*, 550.
on Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, 520.
Leffler, J. A., and Nyström, E.: electric smelting at Trollhättan, Sweden, 173.
Lehigh Coal & Navigation Co.: Coal-mining methods, 698–722.
Leith, C. K.: *Notes on Conservation of Lake Superior Iron Ores*, 231–235.
Leslie Salt Co., San Mateo, Cal.: manufacture of salt from sea water, 940.
Levy, Arthur G., and Howe, Henry M.: *Notes on the Plastic Deformation of Steel During Overstrain*, 532–547, 548, 549.
Lewis, Robert S.: *The Book Cliffs Coal Field, Utah*, 658–678.
Limestone: effect of variation in charge on carbon absorption in steel, 167.
Lindgren, W.: *Discussions: on Asbestos in Southern Quebec*, 962.
on Salt Making by Solar Evaporation, 952.
Linville, C. P.: *Discussion on Notes on the Plastic Deformation of Steel During Overstrain*, 549.
Locomotives (gasoline) in mines: relation to health of miners, 771–778.
Long Beach Salt Co., Long Beach, Cal.: manufacture of salt from sea waer, 940.
Ludlow, Edwin: *Discussion on A Test of Centrifugal Motor-Driven Pumps*, 805.

McCance, A.: theory of hardening of metals, 524.
McKee, Arthur G.: *Discussions: on The Plant of the Duplex Process for Making Steel*, 268.
on Turbo Blowers for Blast-Furnace Blowing, 140.
M. A. N., Nuernberg, Germany: 6,000-hp. coke-oven gas engines, 64.
Maccoun, A. E.: *Discussions: on Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces*, 46–54.
on Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes, 87.
Magnesite: occurrence, preparation and use, 890–900.
Magnesite deposits:
Austria-Hungary, 892.
California, 892, 898.
Greece: island of Euboea, 890.
Norway and Sweden, 896.
Magnetic and non-magnetic conditions of manganese steel, 476–500.
Magnetic qualities of iron-nickel alloys, 480.
Magnetism: electronic theory, 524.

Malleable cast iron: analogy with high-oxygen iron, 358.
Malleable castings: effect of chemical nature of packing material and method of packing, 421.
Manganese: effect on quality of cast iron, 381.
Manganese steel: non-magnetic and magnetic conditions, 476–500.
Manganese Steel and the Allotropic Theory (Sauveur), 501–511; *Discussion* (Burgess), 511–512; (Stead), 511–514.
Manganese-Steel Rails (Hadfield), 327–333; *Discussion* (Hadfield), 335–339; (Potter), 333–335; (Richards), 335.
Manganese Steel, with Especial Reference to the Relation of Physical Properties to Microstructure and Critical Ranges (Potter), 437–470; *Discussion* (Hadfield), 471–475; (Sauveur), 471.
Maps:
Brazil, 144.
Minas Geraes iron-ore deposits, 148.
China: Hong-Kong New Territory iron-ore deposits, 237.
Utah: Book Cliffs and Pleasant Valley coal fields, 662.
Marreco and Morison: investigation of coal-dust explosions, 553.
Means, C. M.: *Discussion on Shot Firing in Coal Mines by Electric Circuit from the Surface*, 735.
Meetings of the Institute: Pittsburgh, Pa., October, 1914, vii.
Metal-cutting tools: action in cutting, 416.
Metal mining: fatal accidents in 1911–12, 765.
Metals:
hardening, 522, 524.
plastic deformation by overstrain, 532–551.
Metropolitan Electric Railway, Paris, France: early use of manganese-steel rails, 330, 333.
Mexico:
asphalt deposits, 865.
oil fields, 859–869.
Michigan:
iron mines: fatal accidents (1901–1911), 765.
iron-ore resources, 200.
Microstructure and critical ranges of manganese steel, 437–475.
Minas Geraes, Brazil:
iron-ore deposits, 143-160.
map, 148.
Mine explosions:
investigations, 552–624.
preventive methods, 567.
Mine taxation, 625–639.
Mine valuation, 188–196.
Minerals of phosphate rock, 901.
Miners: relation of use of gasoline locomotives to health, 771–778.
Mining costs: see Costs.
Mining methods:
asbestos: Georgia, 971.
Quebec, 958.
blasting: Cordeau Bickford (detonant), 738–754.
coal: aerial tramway, Book Cliffs field, Utah, 679, 688.
combined mine car and tramway bucket, 684.
hydraulic filling to minimize subsidence of surface, 878.

Mining Methods:—Continued.
Pittsburgh field, Pa., 654.
protecting gas and oil wells through seams, 870–882.
reworking old territory, 715.
shot firing from the surface, 723–737.
sizes of shaft pillars for different depths from surface, 872.
steep pitch mining of thick veins, 698–722.
iron: safety and welfare work in the Lake Superior region, 755–770.
phosphate rock: Florida, 912.
Tennessee, 920.
Minnesota: iron-ore resources, 200.
Mixers: hot metal:
Bethlehem Steel Co., 258.
Pennsylvania Steel Co., 264.
Tennessee Coal, Iron & Railroad Co., 248.
Moisture as a preventive of coal-dust explosions, 567.
Moldenke, Richard: *Discussions: on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 383–385.
on Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes, 88.
Morganroth, L. C.: *The Occurrence, Preparation and Use of Magnesite*, 890–900.
Morison and Marreco: investigations of coal-dust explosions, [553].
Moss, Sanford A.: expansion of bore of wheels by centrifugal stresses, [107].
Murray, C. B.: *The Need of Uniform Methods of Sampling Lake Superior Iron Ore*, 204–210, 211, 213.
Murray gas cleaner, 49.

Naphthalene: recovery from coke-oven gas, 66.
Need of Uniform Methods of Sampling Lake Superior Iron Ore (Murray), 204–210; *Discussion* (Austin), 215–230; (Ennis), 214; (Kelly), 210–211; (Murray), 211, 213; (Richards), 211; (Sweetser), 211–213.
Neumann lines in manganese steel, 474.
Newfoundland: iron-ore resources, [201].
New Jersey: iron-ore resources, 200.
New Safety Detonating Fuse (Souder), 738–747; *Discussion* (Hall), 747; (Howell), 748–751; (Souder), 751–754.
New York:
asbestos deposit, Tompkinsville, 963.
iron-ore resources, Adirondack region, 200.
salt industry, 934.
Nibecker, Karl: *Discussions: on Notes on Blast-Furnace Operation with a Turbo Blower*, 98, 100.
on Turbo Blowers for Blast-Furnace Blowing, 134–137.
Nickel and iron oxysulphides: penetration of iron by, 349.
Nickel-chromium steel: effect of treatment on magnetic qualities, 484.
Nickel-iron alloys: magnetic qualities, 480.
Nitric acid: preventing corrosion of gas-engine cylinders by, [66].
Nitrogen: influence on quality of cast iron, 344–404.
Nitroglycerin dynamite: rates of detonation, 750.
Norris, R. V.: *Discussions: on The Appraisal of Coal Lands for Taxation*, 631–636.
on Coal-Mine Explosions Caused by Gas or Dust, 622, 624.
on Gasoline Locomotives in Relation to the Health of Miners, 775.
on A Test of Centrifugal Motor-Driven Pumps, 805.

Norway: magnesite deposits, 896.
Notes on Blast-Furnace Operation with a Turbo Blower (VALENTINE), 90–96; *Discussion* (GRAMMER), 102; (JOHNSON); 96, 98, 100, 101; (NIBECKER), 98, 100; (RICHARDS), 97, 101; (STONE), 101; (VALENTINE), 96, 102–103; (WALDO), 99.
Notes on Conservation of Lake Superior Iron Ores (LEITH), 231–235; *Discussion* (LANE), 235.
Notes on an Iron-Ore Deposit near Hong-Kong, China (WELD), 236–245.
Notes on the Plastic Deformation of Steel During Overstrain (HOWE and LEVY), 532–547; *Discussion* (CARPENTER), 547–548; (HIBBARD), 548; (HOWE), 548, 549; (LE CHATELIER), 550; (LINVILLE), 549; (WEBSTER), 549.
Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron (SAUVEUR), 515–517; *Discussion* (BELAIEW), 527; (BENEDICKS), 526; (BURGESS), 520; (CARPENTER), 517–519; (HADFIELD), 521–526; (HOWE), 519; (LE CHATELIER), 520; (SAUVEUR), 528–531.
Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes (FREYN), 56–84; *Discussion* (CHILD), 89; (FREYN), 87–89; (JOHNSON), 87, 88; (LAMB), 84–87; (MACCOUN), 87; (MOLDENKE), 88; (RICHARDS), 88, 89.
Nyström, E., and Leffler, J. A.: electric smelting at Trollhättan, Sweden, 173.

O'BRIEN, T. H.: *Discussion on Shot Firing in Coal Mines by Electric Circuit from the Surface*, 734.
Occurrence, Preparation, and Use of Magnesite (MORGANROTH), 890–897; *Discussion* (DAY), 897–900; (MORGANROTH), 898–900; (SPILSBURY), 899.
Oil and gas: capillary concentration, 829–858.
Oil (petroleum):
accumulation: anticlinal theory, 839.
character changed by migration through the earth, 844, 849.
fractionation: by adsorption, diffusion, or filtration, 844, 849, 853.
illuminating power of oils of various origins, 815.
refining: by distillation, 809.
with liquid sulphur dioxide, 809–828.
with sulphuric acid, 810.
surface tension, 831.
utilization of refinery by-products, 817.
Oil-field waters: chlorides in, 883–889.
Oil Fields of Mexico (ORDONEZ), 859–863; *Discussion* (DAY), 863; (HENRY), 864–867; (WHITE), 867–869.
Oil wells:
Mexico, 863, 868.
through coal seams, 870–882.
Oklahoma: regulation of shot firing in coal mines on Indian lands, 724.
Oliver Iron Mining Co.: pumping plant at Chapin iron mine, 779–808.
Oliver Salt Co., Mount Eden, Cal.: manufacture of salt from sea water, 940.
ORDONEZ, EZEQUIEL: *The Oil Fields of Mexico*, 859–863.
Origin, Mining, and Preparation of Phosphate Rock (SELLARDS), 901–916.
Ovens:
coke: Connellsville field, Pa., 643.
Frick Coke Co., 644.
converter bottom drying, Bethlehem Steel Co., 261.
Overstrain of metals: plastic deformation by, 532–551.

Oxygen:
decarbonization of steel by annealing in, 410.
influence on quality of cast iron, 344–404.
Oxysulphides: iron and nickel: penetration of iron by, 349.

Paint manufacture: barytes as a paint pigment, 983–985.
Panhandle coal district, Pa.: probable life, 645.
Panuco oil field, Mexico, 864.
Pauly, K. A.: *Discussion on A Test of Centrifugal Motor-Driven Pumps*, 801, 803, 807.
Pease, O. D. A., Strand, C. H., and Young, C. D.: *The Heat Treatment of Steel Castings*, 424–431, 435.
Pennsylvania:
coal fields: Pittsburgh, 640–657.
valuation of coal land for taxation, 631.
Pennsylvania Steel Co.: duplex steel plant, 264.
Peters, Richard, Jr.: *Discussion on Coal-Mine Explosions Caused by Gas or Dust*, 624.
Petroleum: see Oil.
Phalen, W. C.: *Salt Making by Solar Evaporation*, 934–950.
Phosphate deposits:
Florida, 907.
Navassa, West Indies, 912.
Tennessee, 911, 917–933.
Western States, 912.
Phosphate rock: origin, mining, and preparation, 901–916, 917–933.
Phosphorus:
content of lithosphere, 905.
effect on quality of cast iron, 378.
Pig-iron production: rate of increase, 646.
Pig Steel from Ore in the Electric Furnace (Keeney), 161–179; *Discussion* (Crawford), 180–187; (Richards), 180.
Piping in steel ingots: Hadfield method of preventing, 270–289.
Pittsburgh Coal Field in Western Pennsylvania (Kuhn), 640–657.
Pittsburgh meeting of the Institute, October, 1914, vii.
Plant of the Duplex Process for Making Steel (Furst), 246–267; *Discussion* (Hibbard), 268; (Johnson), 267; (McKee), 268; (Stoughton), 269.
Portevin, A. M.: decarbonization of steel, [420].
Potash: by-product in California salt industry, [949].
Potter, W. S.: *Manganese Steel, with Especial Reference to the Relation or Physical Properties to Microstructure and Critical Ranges*, 437–470.
Discussions: on Manganese-Steel Rails, 333–335.
on Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel, 496.
Pressure gauge for measuring variations in blast-furnace blowing, 107.
Pumps:
centrifugal motor driven: tests, 779, 808
mine: electrical vs. steam, 801.
centrifugal vs. reciprocating, 802.
cement lining to prevent corrosion, 806.

Quarrying: detonating fuse for blasting, 738–754.
Quarrying Shale by the Tunnel System (Farnham), 974–981; *Discussion* (Day), 981, 982; (Ries), 981, 982; (Ross), 982.
Quebec: asbestos deposits, 954–963.

Rail situation in America, 340–343.
Rails:
causes of failure in service, 316.
comparison of breakage of Bessemer and open hearth, 312, 325.
effect of climatic conditions on failure in service, 310.
finishing temperatures and properties, 302–326.
manganese steel, 327–339.
early use on Metropolitan Electric Railway, Paris, France, 330, 333
manufacture: practice in American mills, 340.
microstructure, 304.
Railroads of Brazil, 155.
Rawdon, H. S., Waltenberg, R. G., Burgess, George K., and Crowe, J. J.: *Finishing Temperatures and Properties of Rails*, 302–306, 322–326.
Read, T. T.: *Discussion on Tin and Coal Deposits of the Fu Chuan District, China*, 696–697.
Reese: origin of phosphate deposits, [905].
Refining Petroleum by Liquefied Sulphur Dioxide (Edeleanu), 809-827; *Discussion* (Bushong), 827.
Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel (Hopkinson and Hadfield), 476–496; *Discussion* (Hadfield), 499; (Hopkinson), 497–499; (Howe), 497; (Potter), 496; (Sauveur), 497.
Reserves of Iron Ore for the United States (Birkinbine), 197–203.
Rice, George S.: *Investigations of Coal-Dust Explosions*, 552–587.
Discussions: on Coal-Mine Explosions Caused by Gas or Dust, 619, 624.
on Gas and Oil Wells through Coal Seams, 870, 875–876, 878, 880–882.
on Gasoline Locomotives in Relation to the Health of Miners, 775.
on A Test of Centrifugal Motor-Driven Pumps, 802, 805.
on Tin and Coal Deposits of the Fu Chuan District, China, 697.
Rice, George S., and Clark, H. H.: *Shot Firing in Coal Mines by Electric Circuit from the Surface*, 723–731, 733, 736–737.
Rice, Richard H.: *Turbo Blowers for Blast-Furnace Blowing*, 104–126, 141–142.
Rice rock-dust explosion barrier, 574.
Richards, J. W.: *Discussions: on Data Pertaining to Gas Cleaning at the Duquesne Blast Furnaces*, 55.
on Manganese Steel Rails, 335.
on The Need of Uniform Methods of Sampling Lake Superior Iron Ore, 211.
on Notes on Blast-Furnace Operation with a Turbo Blower, 97, 101.
on Notes on the Utilization of Coke-Oven and Blast-Furnace Gas for Power Purposes, 88, 89.
on Pig Steel from Ore in the Electric Furnace, 180.
definition of pig steel, [163].
Ries, H.: *Discussions: on Asbestos in Southern Quebec*, 963.
on Quarrying Shale by the Tunnel System, 981, 982.
on Salt Making by Solar Evaporation, 953.
Rock dust as a preventive of coal-dust explosions, 570.
Rocks: porosity of, 829.
Rogers, Austin F.: origin of Tennessee phosphate deposits, [911].
phosphate minerals, [901].
Rolled Steel Roll Shells (Ferguson), 290–301.
Ross, Donald W.: *Discussion on Quarrying Shale by the Tunnel System*, 982.
Rumsey, S. S., and Schwedes, W. F.: *A Test of Centrifugal Motor-Driven Pumps*, 779–801, 807.
Russia: asbestos production, [964].

Safety Movement in the Lake Superior Iron Region (Higgins), 755–770.
Salina Salt Co., Syracuse, N. Y.: manufacture of salt from natural brine, 934.
Sall Mountain asbestos mine, Cleveland, Ga., 969.
Salt Making by Solar Evaporation (Phalen), 934–950; *Discussion* (Day), 951, 952, 953; (Lindgren), 952; (Ries), 953; (Spilsbury), 951–953.
Salt manufacture: German methods, 951.
Salt plugs: precipitation of, 883.
Sampling iron ore: need of uniform methods, 204–230.
Sauveur, Albert: *Manganese Steel and the Allotropic Theory*, 501–511.
Notes on Some Heating and Cooling Curves of Professor Carpenter's Electrolytic Iron, 515–517, 528–531.
Discussions: on Finishing Temperatures and Properties of Rails, 306.
on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements, 389–391.
on Manganese Steel, with Especial Reference to the Relation of Physical Properties to Microstructure and Critical Ranges, 471.
on Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel, 497.
on The Surface Decarbonization of Tool Steel, 420.
Schellenberg, F. Z.: *Discussion on Coal-Mine Explosions Caused by Gas or Dust*, 622.
Schwedes, W. F., and Rumsey, S. S.: *A Test of Centrifugal Motor-Driven Pumps*, 779–801, 807.
Sea water:
density and salinity, on California coast, 943.
production of salt from, 940.
Segregation in steel ingots: Hadfield method of preventing, 270–289.
Sellards, E. H.: *The Origin, Mining and Preparation of Phosphate Rock*, 901–916.
Shale: quarrying by the tunnel system, 974–982.
Shot Firing in Coal Mines by Electric Circuit from the Surface (Rice and Clark), 723–731; *Discussion* (Breth), 731–733; (Clark), 737; (Means), 735; (O'Brien), 734; (Rice), 733, 736–737.
"Skarn" iron ores, Hong-Kong, China, 242.
Smelting (electric) of iron ore, 161–187.
Sodium salts in water of Great Salt Lake, 938.
Solar evaporation process of salt manufacture, 934–953.
Souder, Harrison: *A New Safety Detonating Fuse*, 738–747, 751–754.
Sound Ingots (Hadfield), 270–289.
Spilsbury, E. G.: *Discussions: on Asbestos in Southern Quebec*, 961.
on The Occurrence, Preparation, and Use of Magnesite, 899.
on Salt Making by Solar Evaporation, 951–953.
on The Surface Decarbonization of Tool Steel, 421.
Spring Canyon Coal Co., Storrs, Utah:
aerial tramway, 679–688.
mines and mining methods, 664.
Stag Cañon Fuel Co., Dawson, N. M.: electrical system of shot firing from outside the mine, 734.
Standard Coal Co., Standardville, Utah: mining methods, 668.
Stassano E.: production of steel from ore in the electric furnace, [161].
Stead, J. E.: *Discussion on Manganese Steel and the Allotropic Theory*, 511–514.
Steam: decarbonization of steel by annealing in, 411.
Steam turbines for blast-furnace blowing, 90–142.
Steel:
definitions of the term, 163.

Steel:—Continued.
electric-furnace production, 161–187.
manganese:
allotropic theory, 484, 497, 501–514.
Brinell hardness tests, 510.
decarbonization, 496, 499.
difficulty of forging and rolling, 333, 335.
heat treatment: relation to tensile properties, 446, 505.
heating and cooling curves, 437–444.
magnetic and mechanical tests, 482.
non-magnetic and magnetic conditions, 476–500.
physical properties, 477.
relation of physical qualities to microstructure and critical ranges, 437–475.
slip bands produced by deformation, 474.
structural constituents, 440.
tensile tests, 446, 448, 458, 462, 475, 488.
nickel-chromium: effect of treatment on magnetic qualities, 484.
pig: in the electric furnace, 161–187.
plastic deformation during overstrain, 532–551.
rail: critical ranges, 303.
expansion, 303.
melting and freezing range, 303.
rate of cooling, 303.
rolling temperatures, 302.
reactions in decarbonization, 405.
tool: surface decarbonization, 405–423.
Steel castings:
analyses and physical properties, 425–430, 434.
heat treatment, 424–436.
Steel ingots: Hadfield method of manufacture, 270–289.
Steel plants: duplex process:
Bethlehem Steel Co., 253.
Dominion Iron & Steel Co., 252.
Pennsylvania Steel Co., 264.
Tennessee Coal, Iron & Railroad Co., 246.
Steel-rail manufacture: practice in American mills, 340.
Steel-rail situation in America, 340–343.
Steel rails:
finishing temperatures and properties, 302–326.
manganese, 327–339.
Steel roll shells, 290–301.
Steep Pitch Mining of Thick Coal Veins (Whildin), 698–722.
Stoek, H. H.: *Discussions: on Coal-Mine Explosions Caused by Gas or Dust*, 621.
on A Test of Centrifugal Motor-Driven Pumps, 804.
Stone, George C.: *Discussion on Notes on Blast-Furnace Operation with a Turbo Blower*, 101.
Storey, Oliver W.: *Discussion on The Surface Decarbonization of Tool Steel*, 421.
Stoughton, Bradley: *Discussions: on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 391.
on The Plant of the Duplex Process for Making Steel, 269.
on The Surface Decarbonization of Tool Steel, 420.
definition of steel, [163].

Strand, C. H.: *Discussion on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 387.
Strand, C. H., Young, C. D., and Pease, O. D. A.: *The Heat Treatment of Steel Castings*, 424–431, 435.
Sulphur:
elimination from coke-oven gas, 65.
in pig iron: analytic methods for, 349.
Sulphur dioxide: refining petroleum with, 809–828.
Sulphuric acid: corrosion of gas-engine cylinders: remedies, 65.
Surface Decarbonization of Tool Steel (Emmons), 405–420; *Discussion* (Emmons), 423; (Hofman), 422; (Sauveur), 420; (Spilsbury), 421; (Storey), 421; (Stoughton), 420.
Surface tension of fluids, 830, 837.
Sweden:
iron-ore resources, [201].
magnesite deposits, 896.
Sweetser, Ralph H.: *Discussions: on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements*, 392.
on The Need of Uniform Methods of Sampling Lake Superior Iron Ore, 211–213.
on Notes on the Plastic Deformation of Steel During Overstrain, 549.

Taffanel, J.: *Coal-Dust Explosion Investigations*, 555, 588–593.
Taffanel rock-dust explosion barrier, 574.
Taxation of coal lands, 625–639.
Taylor, S. A.: *Discussions: on The Appraisal of Coal Lands for Taxation*, 637–639.
on Coal-Mine Explosions Caused by Gas or Dust, 623.
on Gasoline Locomotives in Relation to the Health of Miners, 776.
Temperature control of rail rolling, 304.
Tennessee Coal, Iron & Railroad Co.: duplex steel plant, Ensley, Ala., 246.
Tennessee Phosphate Practice (Barr), 917–933.
Test of Centrifugal Motor-Driven Pumps (Rumsey and Schwedes), 779–801; *Discussion* (Hood), 803; (Kelly), 802, 803, 804, 805, 807; (Ludlow), 805; (Norris), 805; (Pauly), 801, 803, 807; (Rumsey and Schwedes), 807; (Stoek), 804; (Thomas), 804–805.
Theisen gas washers at Duquesne furnaces, 22.
Thomas, W. A.: *Discussion on A Test of Centrifugal Motor-Driven Pumps*, 804–805.
Thyssen & Co., Muelheim-Ruhr, Germany: 5,000-hp. coke-oven gas engines, 63.
Tin and Coal Deposits of the Fu Chuan District, China (Yung), 689–696; *Discussion* (Read), 696–697; (Rice), 697.
Tin smelting: Fu Chuan district, China, 694.
Titanium: experiments with, in deoxidation of cast iron, 353.
Tool steel: surface decarbonization, 405–423.
Topila oil field, Mexico, 864.
Tramway line, Book Cliffs coal field, Utah, 679–688.
Trimble, R.: *Discussion on Finishing Temperatures and Properties of Rails*, 307.
Trinitrotoluene blasting fuse, 738–754.
Trinks, W.: *Discussion on Turbo Blowers for Blast-Furnace Blowing*, 137–139.
Tunnel system of quarrying, 974–982.
Turbo blower: blast-furnace operating with, 90–103.
Turbo Blowers for Blast-Furnace Blowing (Rice), 104–126; *Discussion* (Banner), 139; (Braman), 140; (Johnson, W. McA.), 128; (Johnson, J. E., Jr.), 131–134, 140; (Iversen), 128–131; (McKie), 140; (Nibecker), 134–137; (Rice), 141–142; (Trinks), 137–139; (Valentine), 126–128.

United States Bureau of Mines:
experimental mine, Bruceton, Pa., 559.
investigations of coal-dust explosions, 552–587.
safety work in the Lake Superior iron region, 762.
Utah:
coal fields: Book Cliffs, 658–678.
Pleasant Valley, 662.
coal production (1913), 661.
salt industry, 936.
Utah Fuel Co., Carbon County, Utah:
electrical system of shot firing from outside the mine, 723.
mines and mining methods, 661, 677.

Valentine, S. G.: *Notes on Blast-Furnace Operation with a Turbo Blower*, 90–96, 102–103.
Discussion on Turbo Blowers for Blast-Furnace Blowing, 126–128.
Valuation of Iron Mines (Finlay, *Trans.*, xlv, 282); *Discussion* (White), 188–196.
Virginia: iron-ore resources, 200.

Waldo, Leonard: *Discussion on Notes on Blast-Furnace Operation with a Turbo Blower*, 99.
Waltenberg, R. G., Burgess, George K., Crowe, J. J., and Rawdon, H. S.: *Finishing Temperatures and Properties of Rails*, 302–306, 322–326.
Washburne, C. W.: *The Capillary Concentration of Gas and Oil*, 829–842, 850–858.
Discussion on Chlorides in Oil-Field Waters (*Trans.*, xlviii, 687 to 694), 883–889.
Washington: coal deposit, Renton, 982.
Water:
effect on decarbonization of steel, 411.
Great Salt Lake: composition of salts, 938.
production of salt from, 936.
oil field: chlorides in, 883–889.
sea: density and salinity, on California coast, 943.
production of salt from, 940.
Waterhouse, George B.: *Discussion on Finishing Temperatures and Properties of Rails*, 314–316.
Webster, William R.: *Discussions: on Finishing Temperatures and Properties of Rails*, 307–310.
on The Influence on Quality of Cast Iron Exerted by Oxygen, Nitrogen, and Some Other Elements, 387.
on Notes on the Plastic Deformation of Steel During Overstrain, 549.
Weiss, Pierre: magnetic phenomena of metals, 490.
Weld, C. M.: *Notes on an Iron-Ore Deposit near Hong-Kong, China*, 236–245.
West Indies: phosphate deposits, 912.
Westmoreland, Pa. gas-coal basin: probable life, 645.
Wheeler, H. A.: *Discussions: on Asbestos in Southern Quebec*, 963
on The Capillary Concentration of Gas and Oil, 847.
Whildin, W. G.: *Steep Pitch Mining of Thick Coal Veins*, 698–722.
Discussion on Gasoline Locomotives in Relation to the Health of Miners, 776.
White, E. E.: *Discussion on Valuation of Iron Mines* (Finlay, *Trans.*, xlv, 282); 188–196.
White, I. C.: *Discussions: on The Iron Industry in Brazil*, 157–159.
on The Oil Fields of Mexico, 867–869.

Wickhorst, M. H.: *Discussion on Finishing Temperatures and Properties of Rails*, 316.
Williams, Elmer H.: electron theory of magnetism, [492].
Wilson, E. B.: *Discussion on Coal-Mine Explosions Caused by Gas or Dust*, 622.
Wüst: decarbonization of cast iron by iron oxide, 407, 422.

Young, C. M.: *Discussion on Coal-Mine Explosions Caused by Gas or Dust*, 620.
Young, C. D., Pease, O. D. A., and Strand, C. H.: *The Heat Treatment of Steel Castings*, 424–431, 435.
Yung, M. B.: *Tin and Coal Deposits of the Fu Chuan District, China*, 689–696.

Zinc: decomposition by carbon dioxide, 421.